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MINING LAWS OF THE IRISH



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E. P. YOUNGMAN

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MINING LAWS OF THE IRISH FREE STATE



BY

E. P. YOUNGMAN

INFORMATION CIRCULAR

UNITED STATES BUREAU OF MINES

MINING LAWS OF THE IRISH FREE STATE¹

By E. P. Youngman²

PREFATORY NOTE

This paper is one of a series of digests of foreign mining legislation and court decisions that is being prepared in advance of a general report relative to the right of American citizens to explore for minerals and to own and operate mines in various foreign countries. This interpretation of the mining laws of the Irish Free State has been prepared from the best available material in Washington and is released subject to amplification and correction, if necessary, by American foreign-service officials.

INTRODUCTION

This paper is based upon the mines and minerals act, 1931, enacted by the Legislature (Oireachtas) of the Irish Free State (Saorstát Eireann), December 22, 1931, the full title of which is "An act to establish a board to perform functions in relation to mines and minerals, to authorize the making of leases of mines and minerals belonging to or vested in Saorstát Eireann, to make provision for facilitating the working of other mines and minerals and for imposing restrictions on the working of minerals required for the support of buildings and works, to amend the law with respect to mines, and for other purposes incidental to the matters aforesaid."

All citations in this digest, unless otherwise specifically designated, are to the mines and minerals act, 1931.

RIGHTS OF FOREIGNERS

No legal restrictions are placed upon foreigners by the mines and minerals act, 1931, with respect to the right to search for or mine minerals in the Irish Free State.

1 The Bureau of Mines will welcome the reprinting of this paper, provided the following footnote acknowledgment is used: "Reprinted from U. S. Bureau of Mines Information Circular 3750."

2 Rare metals and nonmetals division, U. S. Bureau of Mines.

MINING ADMINISTRATION

General Information

The mines and minerals act, 1931, which with respect to the Irish Free State repeals certain sections (all except 2, 19, 25, and 26) of the mining industry act, 1920, places the administration of mines under the Irish Minister of Industry and Commerce, who (in some instances with the sanction of the Minister of Finance) administers the act of 1931, making regulations thereunder, subject to the approval of the Legislature. The Minister is assisted by the Mining Board (appointed by him), which is a quasi-legal body with some of the powers of the High Court to summon witnesses and command the production of documents, etc. (See section of this paper entitled "The Mining Board.") Section 53 of the 1931 act repeals for the Free State section 1 of the 1920 act, which set up for the United Kingdom a Mining Department under the British Board of Trade.

As sections 19 and 25 of the act of 1920 have not been repealed, it may be assumed that with respect to actual mining regulations the British coal mines act, 1911, and the metalliferous mines regulations acts, 1872 and 1875, are still in force in the Free State. This assumption is strengthened by the fact that, although by section 54 of the mines and minerals act, 1931, the Minister may make regulations "in respect of any matter or thing referred to in this act as prescribed or to be prescribed," the act does not treat actual mining regulations except with respect to inspection and to the fencing of abandoned mines. (Sec. 53, act of 1931; sec. 19 and 25, act of 1920.)

The Mining Board

The act of 1931 established a Mining Board (of three members), appointed by the Minister of Industry and Commerce. The Minister shall determine the period during which each member shall serve, shall remove any member at his discretion, shall appoint substitutes to act temporarily, and shall make regulations with respect to the sittings and the procedure of the Board. (Sec. 6 and 7.)

Qualifications of members.--The Chairman of the Board shall be a practicing barrister or solicitor of at least 10 years' standing; one member shall be a member of the panel of official arbitrators appointed under the acquisition of land (assessment of compensation) act, 1919, as amended by the acquisition of land (reference committee) act, 1925 (No. 22 of 1925); and the other member shall be an officer of the Minister. (Sec. 6.)

Powers.--The Board shall have the powers vested in the High Court or in a judge thereof with respect to compelling the attendance of witnesses, examining them on oath or otherwise, and compelling the production of documents.

The Board may cite for contempt of court any person doing anything that would be contempt of court were the Board a court of justice having the power to commit for contempt. (Sec. 8.)

The Board or any member or officer thereof may enter any land for any purpose in connection with the functions and duties of the Board--A fine not to exceed £10 to be imposed for obstructing or impeding any member or officer of the Board in his right of entrance. (Sec. 9.)

OWNERSHIP OF MINERALS

Through the Irish Land Act, 1923, and the repeal or modification of various sections of earlier land acts, by section 2 of the mines and minerals act, 1931, all mineral substances except stone, gravel, sand, or clay belong to the Irish Free State.

Subsection 5 of section 45, Irish Land Act, 1923, with the first proviso, which is still in force, reads as follows:

On the vesting of any land under this act, the exclusive right of mining and taking minerals and digging and searching for minerals on or under that land, including such mineral rights as may be superior interests, shall vest in Saorstát Éireann:

Provided that this subsection shall not apply (a) to any mine or quarry that is being worked or developed by the owner or his lessee: or (b) to any stone, gravel, sand, or clay.

The second proviso, which was repealed, reads as follows:

Provided also that where any such rights vested in Saorstát Éireann under this subsection are at any time hereafter let, leased, or sold, the person who would have been entitled thereto, if they had not so vested, shall be entitled to receive such share of any rent, purchase money, or any other net profit received in respect to the same as shall hereafter be determined by the Oireachtas (Legislature).

By substituting the "Saorstát Éireann" for the "Irish Land Commission" in subsection 3 of section 13 of the Irish Land Act, 1903, the Government has the following right (subsec. 1, sec. 2):

On the sale under the land purchase acts of any land by the Land Commission or of any land comprised in an estate by the owner of the estate, there shall be reserved, in the prescribed manner, to Saorstát Éireann the exclusive right of mining and taking minerals and digging and searching for minerals on or under that land.

Provided that this subsection shall not apply--

- (a) To any demesne or other land resold in pursuance of section 3 of this act.
- (b) To any such right that constitutes a superior interest or that is vested in the Crown.
- (c) Or to any stone, gravel, sand, or clay.

With respect to any letting, lease, sale, or demise of any exclusive right of mining and taking minerals (reserved to the Irish Land Commission by section 13 of the Irish Land Act of 1903) made by the Commission before the passage of the 1931 law under section 1 of the Irish Land Act, 1907, as amended by section 37 of the Irish Land Act, 1909, the new law shall not be effective.

Questions with respect to exclusive mining rights may be referred by the Minister to the Irish Land Commission, exclusive of the Judicial Commissioner. An appeal from the Commission's decision shall lie with the Supreme Court. The rules regulating all such procedure may be made by the Irish Land Commission. (Sec. 4.)

Such questions include the following (sec. 4):

1. Whether an exclusive mining right has before the passing of the act of 1931 been reserved to the Land Commission under Section 13 of the Irish Land Act, 1903, and if so reserved whether the right has been purchased or disposed of by the Commission in accordance with law, and if so disposed what the nature and extent of such disposition are.

2. Whether an exclusive mining right has on or after the passing of the act of 1931 been vested in Saorstát Éireann under section 13 of the land act of 1903, as amended by the 1931 act.

3. Whether an exclusive mining right has been vested in Saorstát Éireann under section 45 of the land act of 1923.

4. Whether any particular substance "can" be mined and taken under any particular mining right mentioned in the foregoing paragraphs.

PROTECTION OF STATE LANDS

The Minister shall not lease any "State mines or minerals"³ or exclusive State mining right,⁴ nor shall he exercise or authorize another to exercise the right of entrance for experimentation (sec. 15):

3 Mines and minerals that at the date of the passage of the 1931 act belong, by virtue of article 11 of the Constitution, or otherwise, to the Irish Free State; also those that since the passage of the act have or shall become vested in the Free State. (Sec. 5) The word "minerals" includes all minerals and substances in or under land obtainable by underground or surface working. (Sec. 5.)

4 Any exclusive mining right that immediately before the passage of the 1931 act belonged to or was vested in the Free State or was reserved to the Irish Land Commission; and any exclusive mining right that on or after the passage of the act became or becomes vested in the Free State. (Sec. 5.)

1. Upon lands to which the State Lands Act, 1924 (No. 45 of 1924) applies, except after consultation with the Minister for Finance.
2. Upon lands to which the State Lands (Workhouses) Act, 1930 (No. 9 of 1930), applies, except after consultation with the Minister for Local Government and Public Health.
3. Upon lands vested in or in the occupation of the minister head of a department of State, except after consultation with such minister.

The Minister shall make no order (with respect to the section of the law entitled "Grant of Rights in Non-State Mines and Minerals") in relation to any minerals in or under any land vested or in the occupation of the minister head of a department of State, except after consultation with such minister. (Sec. 26.)

STATE MINES AND MINERALS

Mining Rights

General decisions.--The Minister of Industry and Commerce shall decide whether it is in the interest of the public to grant any State mines or minerals or any exclusive State mining right to any person by way of a lease, for a term not to exceed 99 years.

The demise of State mines, minerals, or exclusive State mining rights may be made by way of a takenote (prospecting lease) for a period not to exceed two years, with the option to the holder thereof to succeed to a lease for such term as will with the term of the takenote, or prospecting lease, not exceed 99 years.

Every lease shall contain any covenants, conditions, or agreements (except with respect to a renewal) as the Minister shall consider proper and to which the lessee shall have agreed. (Sec. 11.)

A lease confers upon the lessee the right to enter upon the land included in the lease, subject to the payment of the required compensation. (see section of this paper entitled "Compensation.") (Sec. 12.)

Royalties, etc.--The Minister of Industry and Commerce and the Minister for Finance shall decide whether in the public interest a lease should be free from any payment or whether a fine or other preliminary payment, or rent (including royalty rent, variable according to the price or the value of the minerals won), or both, should be paid. (Sec. 11.)

Returns.--Every lessee shall furnish to the Minister, in the prescribed form, within 28 days of being requested so to do, such information as the Minister may require with respect to the minerals and the working thereof on the leased land. Failure or refusal to furnish the information desired or furnishing false information shall make the lessee liable to a maximum penalty

of £10 and in the case of a continuing offense to a further fine (not to exceed £10) for every day of default. (Sec. 16.)

Lessor's right of entrance.--The Minister (or representative) may, for the purpose of ascertaining the value of any State mines, minerals, or exclusive State mining right, enter the land concerned and make such experiments (including borings) as may appear to him to be necessary. This right of entrance shall not be exercised unless one month's notice has been given to the occupier of the land. (Sec. 13.)

NON-STATE MINES AND MINERALS

Mining Rights

In general, no mining right nor any auxiliary right shall be granted by the Minister with respect to non-State mines or minerals unless it is shown that it has not been practicable to obtain it by private arrangement. (Sec. 20.) Mining rights in such mines or minerals may be granted by the Minister when it seems that the minerals may be left permanently unworked for various reasons, when they are not being worked efficiently, or when boundary adjustment agreements are not effective. Such rights are classified as follows (sec. 18):

1. A mining (proprietor's) right, where minerals are comprised in land subject to a lease, exception, reservation, restriction, covenant, or condition or are otherwise incapable of being worked without the concurrence of two or more persons and where some person interested in the minerals desires to work them himself or through his lessees.

2. A mining (small parcels) right, where minerals are owned in such small lots that they cannot be conveniently worked alone and some person having interest therein desires (by himself or through others) to work them with adjacent minerals.

3. A mining (area) right, where the minerals cannot be worked economically except as a whole throughout the area and where a person that has an interest in part of the minerals desires to work (by himself or through others) all the minerals of the area.

4. A mining (unworked minerals) right, where the minerals are not being worked or are being inefficiently worked and no satisfactory reason can be given therefor by the person interested, and where some other person desires to work them (by himself or through another).

5. A mining (adjustment of boundaries) right, where a mining operator has entered into an agreement with an adjoining operator for the adjustment of boundaries between mines in order to reduce the amount of unworked minerals or to enable the minerals in the mines to be worked more efficiently and the agreement is of no effect because of the refusal of the lessors of such mines or of the owners of the surface to concur therein.

Any of the non-State mining rights, except the "unworked minerals right," may be granted with respect to minerals comprised in a State mines lease for a term not to exceed the unexpired term of that lease. (Sec. 18.)

Applications.--An application for a mining right, which shall be made to the Minister, shall be in the form prescribed and shall include a statement of the circumstances that would justify the grant of the right applied for. The applicant shall serve copies of the application upon the prescribed persons, who may make whatever representations they may desire to the Minister. Three weeks after the receipt of an application, the Minister shall consider, after making any inquiries he thinks fit, whether the applicant has established a prima facie case; and unless he decides that such a case has not been established, he shall refer the application to the Mining Board for its report. Upon receipt of the Board's report, he may grant the right applied for (if he is satisfied that to grant such a right is in the public interest), subject to such conditions and for such a period as he thinks fit, taking into consideration the time reasonably necessary fully to work the minerals. When an applicant's interest is a leasehold interest, due regard shall be had for that interest. (Sec. 21, 22, and 23.)

Should separate applications for the working of the same minerals be made, the Minister shall report the circumstance to the Board for its report. Upon receipt of the Board's report, the Minister shall decide to which applicant (if any) the right should be granted or whether the right to work only part of the minerals should be given to one applicant and the right to work another part to another applicant. The Board, in its report, shall take into consideration how the minerals can be most conveniently worked. The Minister shall take into consideration the Board's report, the respective rights of the applicants in the surface or adjacent minerals, and generally all the circumstances of the case. (Sec. 24.)

Revocation.--The Minister may revoke a mining right whenever he is satisfied that the holder thereof is not efficiently working the minerals covered by the right. In case of revocation, no part of any compensation paid shall be refunded. (Sec. 25.)

Returns.--Every grantee of a mining right shall furnish to the Minister, within 28 days of the request therefor, any information the Minister may require with respect to the minerals under the right. The penalty for failure or refusal to furnish the desired information or for furnishing false information is a fine not to exceed £10 and in the case of a continuing offense a further fine not to exceed £10 for each day of offense. (Sec. 27.)

Ancillary Rights

Ancillary rights may be granted by the Minister to persons holding mining rights. An ancillary right may not be given in, through, or over State mines and minerals concerning which an exclusive State right to mine is exercisable, unless it is covered by a lease, and then only for the unexpired term of that lease.

No ancillary right shall be granted until it is proved that it is not reasonably practicable to obtain such right through private arrangement, because the persons with power to grant it have conflicting interests, cannot be found, have not the necessary power of disposition, or either refuse outright or make unreasonable terms. (Sec. 19 and 20.)

Nature of rights.--In general, an ancillary right is any facility, right, or privilege required in order to work minerals properly and conveniently. Some of such rights are enumerated as follows (Sec. 19):

1. Right to let down the surface.
2. Right of airway, shaftway, or surface or underground wayleave, or other right for the purpose of access to or conveyance of minerals or machinery or the ventilation or drainage or working of the mines.
3. Right to use and occupy the surface for the erection of crushing and dressing mills, washeries, coke ovens, railways, aerial ropeways, aerial tramways, by-product works, brickmaking or other works, or dwellings for employees.
4. Right to a supply of water or other substances in connection with the working of minerals.
5. Right to dispose of water or other liquid matter obtained from mines or any by-product works.
6. Right to dispose of any waste products obtained in connection with the working of minerals.
7. Right to dam or divert any river or watercourse, including an artificial watercourse.
8. Right to divert sewers, water mains, and pipes.
9. Right to divert a public road, street, or way or a private way and to substitute for an existing bridge another bridge on a different site.
10. Right to divert a railway or tramway.
11. Right to demolish buildings that impede the proper working of mines and minerals.

Application.--The procedure to be followed in connection with an application for an ancillary right is practicably the same as that for applications for mining rights in non-State mines and minerals. (Sec. 24.) (See p. 13.)

Consideration of right.--When the ancillary right applied for is that of letting down the surface, the Minister shall weigh the relative value of

the buildings erected or to be erected. If no works or buildings exist, the Minister shall consider the extent to which the intended use of the surface would be affected by subsidence. The Minister must consider in all such cases whether the working of the minerals is more or less important to the public interest than the buildings or the surface affected; to what extent, if any, retention of the minerals is required for the protection of any mines or other works from flooding or for any other purpose; and as far as is relevant the royalties, covenants, and conditions reserved by or contained in the applicant's lease or those customarily included in leases in the district. (Sec. 23.)

SUBSIDENCE

Any person having any interest in any land may make application to the Minister for the placing of such restrictions on the working of minerals under or adjacent to that land as he may consider necessary to support sufficiently all buildings or works, provided that he is not able to obtain that support through private arrangement, because the persons with power to grant such a right have conflicting interests, cannot be found, or have not the necessary powers of disposition (by reason of defect of title, legal disability, or otherwise), or are not willing. (Sec. 28.) When the Minister has made an order imposing a restriction (with reference to the quantity or position of minerals or methods of working or packing), the enforcing of that order is vested in the person making the application therefor, who shall be liable for the payment of compensation. (Sec. 30.) No such restriction shall be imposed with reference to State mines and minerals, or minerals with respect to which an exclusive State mining right is exercisable, unless they are under a mining lease. (Sec. 30.)

Every application for restrictions upon the working of minerals shall be made to the Minister, shall justify the asking, and shall be dealt with as are applications for mining rights. (Sec. 28, 29, and 30.) (See pp. 13 and 14 of this paper.)

COMPENSATION

General provisions.--The Minister may make the payment of compensation a condition of the granting of a right in non-State mines or minerals or of granting an order imposing a restriction with respect to minerals required for support. (Sec. 23, 30, and 33.)

Both the lessee of State mines or minerals (in the exercise of his mining right) and the lessor (in the exercise of his right of entry for experimenting) are subject to the payment of compensation. Whenever damage is done to the surface of any land or any buildings on any land in the exercise of the right of entry, or of the right of entry for experimenting, or of an exclusive State mining right to mine or search for and take State mines and minerals, the person exercising any such right shall be liable to pay compensation for damage. (Sec. 12, 13, and 14.)

The Board, in assessing compensation (which it does in default of agreement), shall be guided by what would be fair and reasonable between a willing

grantor and a willing grantee. (Sec. 39 and 47.) A decision of the Board shall be final; but the Board may and shall, if the High Court so directs, state, in the form of a special case, any question of law arising in the course of the proceedings and may state its award as to the whole or part thereof in the form of a special case for the opinion of the High Court. The decision of this court shall be final and not subject to appeal to the Supreme Court. (Sec. 49.)

Provisions with respect to land the subject of a land purchase annuity⁵-- In connection with either State mines or non-State mines or minerals, if the amount of compensation is fixed by agreement and the land is subject to a land-purchase annuity, the following conditions shall have effect (sec. 40):

1. The Irish Land Commission shall be a party to the agreement.
2. If the Irish Land Commission is not a party, "the payment of the amount of compensation fixed thereby shall not be a good discharge to the person paying it of his liability to pay such compensation."
3. Any such compensation or part thereof that is payable to the Irish Land Commission under such agreement shall when received be applied by the Commission in reduction of any arrears due on foot of such annuity, or if there are no arrears or they are less than the amount of such compensation, then in or towards the redemption of the purchase annuity.

Application--Any person claiming to be entitled to compensation (the Irish Land Commission in case land in question is subject to a land-purchase annuity) shall apply to the Board for determination of the amounts of the compensation due. (Sec. 41.)

Award--The Board, after inquiring into an application, shall make an award, containing the following matters: (a) A statement of the amount; (b) the names of the persons by whom and to whom compensation is to be paid; (c) the allocation of the award if it is paid to two or more persons.

When the compensation to be assessed is in connection with land that is subject to a land-purchase annuity and the Irish Land Commission so requests, the award shall (1) order payment of the whole or of a part to the Irish Land Commission if the amount of such compensation is equal to or less than the redemption price of such annuity and the arrears (if any); (2) order payment to the Land Commission of such sum, not exceeding the amount of the redemption price and arrears, if the amount of compensation exceeds the redemption price of the annuity and the arrears. (Sec. 41.)

Every award shall be signed by two members of the Board; it shall be conclusive evidence of all matters purported to be certified therein; any sum directed by an award to be payable by one person to another shall be a debt and shall be recoverable as a simple contract debt. (Sec. 42.)

Fees--Any applicant other than the Land Commission shall pay whatever the Minister (with the sanction of the Minister for Finance) shall prescribe. (Sec. 44)

⁵ Payable under the land purchase acts to the Irish Land Commission. (Sec. 1.)

Costs and expenses.--The Minister may order all or a part of the expenses incurred by the Board in connection with an application for compensation to be paid by the applicant, unless such applicant is the Irish Land Commission. Any amount certified by the Minister shall be recoverable as a simple contract debt. The Minister may require security for the costs and expenses from the applicant. (Sec. 44.)

The costs (including fees, charges, and expenses of the award) of the parties to a compensation proceeding shall be at the discretion of the Board, which may direct to whom, by whom, and the manner in which they may be paid; it may disallow the costs of counsel. The Board may itself tax the amount of the costs or direct in what manner they shall be taxed. If the applicant for compensation has been ordered to pay all or part of the costs, the person by whom the compensation is payable may deduct the amount of such costs from the compensation. (Sec. 46.)

Right of audience before the Board.--The following persons shall be entitled to be heard when compensation is being assessed: (1) Any claimant of any estate or interest in the land; (2) the person liable to pay the compensation; and (3) the Irish Land Commission if the land in question is subject to a land-purchase annuity. (Sec. 45.)

REGULATION OF MINES

In general, the Minister may make regulations with respect to anything referred to in the mines and minerals act of 1931. Every regulation shall be laid before each House of the Legislature; if an annulling resolution is passed by either House within the next subsequent 21 days during which such House has sat after the regulation is presented to it, the resolution shall be annulled, without prejudice to the validity of anything previously done under it. (Sec. 54.) (See also section of this paper entitled "Introduction.")

Inspection.--Any one proposing to sink a shaft or borehole to more than 30 feet below the surface shall give to the Minister written notice of his intention; he shall keep a journal with respect to his work and shall retain for a period of not less than six months such specimens of the strata passed through as may have been obtained as either cores or fragments.

A Government inspector, who shall have the right at all reasonable times of free access to any such shaft, borehole, or core, shall have the right to inspect journals and specimens and to take copies of journals and representative specimens. However, if the person sinking the shaft makes written request to the Minister to treat as confidential a journal or a specimen, the Minister shall comply with the request unless he considers it unreasonable. (Sec. 50.)

Inspectors shall have the right, at all reasonable times, of free access to all underground workings and to any information or specimens of strata he may reasonably require. (Sec. 51.)

Fencing.--Before the abandonment or discontinuance of any State mines or the exercise of any exclusive State mining right, the Minister may cause the top or entrance of every shaft or outlet to be fenced by a permanent structure. (Sec. 52.)

MISCELLANEOUS PROVISIONS

The following additional provisions are applicable to the sections of the law relating to mining rights in non-State minerals and to subsidence restrictions.

General.--The Minister, in granting a mining right for non-State minerals or a restriction upon the working of minerals required for support, shall not "confer any greater or other power than would be conferred if such right had been granted by a person legally entitled to grant such a right or relieve the grantee from any obligation or liability to which he would have been subject had such right been granted by any such person." (Sec. 35.)

Life tenants or trustees.--An order may confer rights on a tenant for life, any trustee, personal representative, or other person in a fiduciary capacity, which rights shall be deemed to form part of the property subject to the settlement of the estate of the deceased person or the property subject to the trust, as the case may be. (Sec. 34.)

Fees.--Fees for applications shall be such as the Minister, with the sanction of the Minister of Finance, shall prescribe. (Sec. 31.)

Right of audience at Board hearings.--At an inquiry held by the Board, the following persons shall be entitled to be heard (sec. 32):

1. The applicant or applicants.
2. Any person claiming to have any estate or interest in the minerals subject to the application.
3. Any person that would be substantially affected by the grant, in the opinion of the Board.
4. The Minister.

Costs and expenses of inquiries before the Mining Board.--The Minister may order an applicant to pay all or any of the costs or expenses incurred by the Board in connection with an inquiry instituted with respect to an application--the amount to be recoverable as a simple contract debt; or the Minister may require an applicant to furnish security for the costs of an inquiry. (Sec. 36.)

The Board, if it considers it just that any person or persons attending an inquiry should pay all or part of the costs, shall make a recommendation to the Minister to that effect; and the Minister shall issue the proper orders (the amounts to be paid not to exceed those recommended by the Board). (Sec. 37.)

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UNITED STATES BUREAU OF MINES
SCOTT TURNER, DIRECTOR

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INFORMATION CIRCULAR

ONYX MARBLE AND TRAVERTINE



BY

OLIVER BOWLES AND D. M. BANKS

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UNITED STATES BUREAU OF MINES

ONYX MARBLE AND TRAVERTINE¹

By Oliver Bowles² and D. M. Banks³

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2 Supervising engineer, building materials section, U. S. Bureau of Mines.

3 Building materials section, U. S. Bureau of Mines.

INTRODUCTION

Calcareous rocks are used extensively for building and ornamental purposes. The principal varieties are marble and limestone. Onyx and travertine are subdivisions of these groups that are sufficiently distinctive in origin and use to merit separate classification. The purpose of this paper is to define the terms "onyx" and "travertine" and to describe the occurrences and uses of the materials bearing these names.

ORIGIN AND CHARACTER

Confusion exists in the use of the word "onyx." True onyx is a banded chalcedony, a form of silicon dioxide related to jasper, agate, and flint. Calcium carbonate is commonly deposited from solution in successive layers of varying color closely resembling onyx, and for this reason such calcareous rock is also called "onyx." To distinguish the calcareous from the siliceous varieties, the former is called "onyx marble." It is sometimes designated "Mexican onyx" because famous deposits are situated in Mexico.

Onyx marble and travertine are described together because they are of similar origin. Calcium carbonate is practically insoluble in pure water but is soluble to some extent in water charged with carbon dioxide. Thus, carbonated springs may carry calcareous solutions. When they reach the surface the carbon dioxide is dissipated in the atmosphere, and, the solvent being lost, the calcium carbonate is deposited. Many deposits are found to consist of aragonite rather than calcite. Impurities, such as iron and manganese oxides, in the solution may vary during deposition from time to time, and consequently successive layers may be of different colors.

Onyx marble is regarded generally as a deposit from cold-water solutions. It may be formed at the surface of the earth around exits of springs or in rifts, cracks, or cavities in the rock through which the solution flows. Deposits may also take the form of stalactites, stalagnites, and surface coatings in limestone caves. For this reason the mineral sometimes is called "cave onyx." Onyx takes a high polish and, therefore, is classed with the marbles. Its translucence and attractive banding fit it for highly decorative uses.

Travertine originates in much the same way as onyx, but it is regarded as a product of precipitation from hot springs. Sometimes deposition takes place on the bottoms of shallow pools where the rate of escape of the carbon dioxide gas is retarded sufficiently to permit the material to assume a compact and crystalline texture. Travertine is characterized by the presence of numerous irregular cavities varying from the size of a pin's head, or smaller, to a half inch or more across. Most travertines are not susceptible of high polish and therefore are classed with the limestones rather than with the marbles. A variety too soft and porous for building purposes is known as "calcareous tufa," and deposits of fine unconsolidated particles mixed with clay and other impurities are termed "marl."

ONYX MARBLE

Occurrence

Onyx marbles occur in many limestone districts of the United States and foreign countries, but only a few localities produce material of a quality and quantity to justify commercial production. The deposits now producing or that are noteworthy are here described.

Foreign Countries

Mexico.--The Mexican onyx deposits are the most important commercially at the present time. The quarries, known as the New Pedrara, are at El Mar-mol, Lower California, 51 miles inland from Santa Catarina, the port of shipment, and about 330 miles southeast of San Diego, Calif. The largest of them are on a mesa about 40 feet high, 3,000 feet long, and 1,200 feet wide. Commercial onyx occurs in three beds. The upper bed is thin and highly colored and provides relatively small masses suitable for novelty work such as automobile gear-shift balls, pen bases, lamp fittings, ash trays, book ends, and candlesticks. The second bed is 1 to 2 feet thick and furnishes both novelty and block onyx, and the third or lowest stratum supplies large blocks ranging from 1 to 4 feet in thickness that may be cut into sound slabs for decorative uses in banks, theaters, and hotel lobbies. Explosives are used only in stripping, and blocks are separated in the quarry by plug-and-feather wedging. Annual production is about 25,000 cubic feet, and a reserve of 1,500,000 cubic feet is said to be in sight. However, production has been much smaller since 1930.

The most difficult problem is transportation, for blocks must be hauled in 5-ton trucks with trailers 51 miles to Santa Catarina where storage yards are maintained. Stiff-leg derricks on a 400-foot wharf load the blocks on lighters which carry them to ships lying beyond the breakers. From there they are conveyed to San Diego where they are manufactured into finished products or held in bonded yards for shipment to many countries.

The Mexican product is very beautiful owing to its translucency, freedom from flaws, and fine color banding. The colors are peculiarly delicate, and the quality is remarkably uniform. Light greens, pearl white, and delicate rose are the more common colors, all variegated by a network of fine sharp veins of rose red.

Other districts in Mexico have produced onyx, but the deposits are not commercially important at present. A quarry near the city of Puebla, the famous La Pedrera, many years ago furnished a fine quality of green onyx ranging from a very light to a very dark tint showing a slight dash of red or pink. The high reputation of this stone was recognized the world over.

Three quarries of some importance in the vicinity of Tehuacan, Puebla, have furnished onyx of variegated colors and of pure white.

From several quarries on the Tequisistlan River northwest of Salina Cruz in Oaxaca onyx of excellent quality was exported to the United States for a number of years, but owing to poor methods of cutting the blocks and to competition with the cheaper onyx from Lower California the industry did not prosper.

Argentina.--Translucent onyx marble with attractive green, brown, and red veins occurs at several points in Argentina. It is sometimes called "Brazilian onyx." A small amount of the exports from this country now reaches the American market.

Chile.--Onyx production has been reported from the province of Atacama, Chile.

France.--Onyx marbles in France are confined to the slopes of the Pyrenees. The most noteworthy is "Stalactite du Bedat," quarried at Hautes-Pyrenees.

Germany.--In Hanover, Germany, an onyxlike mineral locally called "onysette" is being worked in a small way to produce blocks which are manufactured in Germany and Holland.

Italy.--Many localities capable of affording fine onyx marble exist in different parts of Italy. Yellow, white, and dark-brown varieties have been quarried in the Provinces of Cuneo, Como, Bergamo, and Vrescia. The Appennine Range has furnished a great deal of material for the churches and palaces of Italy.

Spain.--In Spain a wax-yellow onyx occurs in caverns at Malaga, and white and green varieties at Arcenia in Andalusia.

Algeria.--Celebrated deposits of onyx occur in the northern part of Algeria in the Province of Oran. These deposits were important sources of supply for the Romans. The stone varies greatly in color and shade, ranging from pure white to rose, bright red, golden yellow, and, more rarely, green.

Egypt.--Egypt was one of the most important sources of onyx marble in ancient times. Unlike that of Algeria the product is a stalagmitic type. Many deposits were worked in the Nile Valley as early as the time of the First Dynasty, but most of them have been worked out or have been abandoned for other reasons. Recent supplies have been obtained from near the town of Assuit, the site of ancient Lycopolis.

Persia.--Deposits in Persia which furnished the beautiful onyx found in the old buildings and tombs of that country were located near Tabriz on the eastern shore of Lake Oroomiah and in the mountains near Yezd. The stone is of a pink, greenish, or milk-white color, streaked with reddish or copper-colored veins.

United States

Arizona.--Several deposits of onyx marble of commercial importance are known in the United States. A deposit that has been operated periodically for 40 years is at Mayer in Yavapai County, Ariz., 15 miles southeast of Prescott. About 10 quarries have been operated within the past few years.

The deposit varies in thickness from a few inches to 25 feet and covers an area of about a square mile. It consists of bedded calcite and aragonite beautifully colored by iron oxides. Onyx in a great variety of colors is obtainable. The green variety with dashes of red and brown known as "Paisley shawl" is a compact highly lustrous stone and is especially desirable. The onyx is quarried in large slabs and shipped to a finishing plant at Dyersville, Iowa.

Deposits in the extreme southern part of the State in Maricopa County on Camp Creek west of Cave Creek are also quarried. They consist of boulders of calcite and aragonite in soft travertine. The prevailing colors are green and yellow with veins of brown and red. The boulders are cemented together with plaster of paris, sawed into blocks, and then polished at the finishing plant of the company located at Phoenix.

A deposit in Santa Cruz County about 4½ miles south of Greaterville, Prince County, is reported to consist of material so free from cracks that the size of blocks obtainable is limited only by the equipment that may be installed to handle the stone and by transportation facilities. It comes in brown of various shades.

California.--Quite a number of deposits of onyx marble in California have been reported, but most of them are too small to be of commercial importance.

A quarry at Suisun City in Solano County, which has recently been producing, furnishes a stone varying from light amber to deep resinous brown.

A light-brown beautifully veined variety has been reported also from San Bernardino County.

A deposit which furnished material in considerable quantities about 25 years ago is located in San Luis Obispo County near Musick. The colors of this deposit are quite variable. The stone is close in texture, is translucent, and takes a high polish. Blocks 3 to 6 feet square were taken from the quarries. Specimens of the material are on exhibit in the Museum of the California State Mining Bureau.

A small deposit of a rich deep-brown color with veins of lighter yellowish brown has been worked on Sulphur Creek, Colusa County.

Montana.--A vein of onyx 65 feet wide occurring about 5 miles north of Manhattan, Gallatin County, has been developed to some extent. A silicified banded ornamental rock, known locally as "Montana onyx", occurs near Virginia City, Madison County.

New Mexico.--Onyx marble has been produced near Rio Puerco in Valencia County, N. Mex.

South Dakota.--A deposit estimated to be 300 feet wide and 1,500 feet long is reported in the Black Hills of South Dakota near Custer.

Utah.--In Utah quarrying has taken place at Pelican Point on the west shore of Utah Lake. The stone of dark and light amber shades has been described as being beautifully translucent and with colors of great depth and brilliancy. It was used in the interior of the City and County Building, Salt Lake City. A deposit at Lehi in the same county furnished bright lemon and orange, dark buff, chrome yellow, and white slabs.

A deposit in Tooele County ranging in width up to 300 feet and in length to 4,500 feet also has been quarried. The colors of the material range from flaky whites through creams, pinks, and lavenders to yellows shaded with brown. The stone was used in the State Capitol in Salt Lake City.

Other States.--Onyx marbles also occur in the Eastern Appalachian region of the United States, especially in the cave districts of Virginia. It has sometimes happened that these deposits are of sufficient extent to warrant the opening of small quarries, though the stone is rarely of sufficient beauty to enter into active competition with the onyx marbles of Mexico, Arizona, and California. A quarry was at one time developed at Willowton, W. Va., and produced material of a greenish-white color similar to the Mexican onyx.

Several cave deposits in the vicinity of Cave City and Mammoth Cave, Ky., yielded small quantities of material for a number of years. Quarries in the broken-down caves of eastern Tennessee, southwestern Missouri, and Arkansas also have been opened and worked for a limited period, some of which furnished a fair grade of material.

Uses

Onyx marble at present is used principally for interior decoration and for novelties. In the gift-shop trade small pieces may be found in the form of ink stands, lamp bases, smoking sets, clock cases, paper weights, card receivers, and plaques. Table tops and other pieces of furniture are sometimes made of onyx. A prominent use is for gear-shift balls. Larger sizes are employed in staircases, for mantels and fireplaces, or as moldings or columns; large slabs are used for ornamental wainscoting in hotel lobbies, banks, and theaters, and also as panels fronting soda fountains. One writer has deplored the fact that so beautiful a stone is used in modern times largely for flat surfaces only, all brilliantly polished, "yet the stone will cut to as sharp an edge as the finest Carrara marble and is eminently adapted for bas-relief, small statues, busts, and objects of like nature."

In Egypt during very early times onyx marbles were used for making small articles such as jugs, bowls, vases, and urns of different shapes and designs. These were used for domestic purposes and also in religious rites. Few, if

any, of these articles were polished, though many show great skill in workmanship. Statuettes of that period carved from onyx may be found in museums. This type of marble, which was available in slabs 20 feet long, was used also for the decoration of tombs and temples and for construction of shrines. Large blocks or columns of the material were also used in buildings. One of the finest pieces of work now on exhibition in a museum in London is an elaborately carved sarcophagus hollowed out of a single piece of onyx marble.

The stone was used by the Romans in much the same way. In an Arabian mosque built during the twelfth century was found the remains of an old court flagged with onyx. References are made to the use of onyx "in the palmy days of the Persian empire" in the form of slabs to pave fountains and baths. Small pieces were used in windows and doors instead of glass.

Onyx marble was quarried in Mexico before the time of the Spaniards. The Aztecs used it extensively in the ornamentation of their temples; altars and baptismal fonts were made of it; thin slabs constituted the window panes.

Production and Imports

So little onyx marble is quarried in the United States that it has never been possible to publish figures of production separate from those of marble. The amount produced never has been large.

Import figures likewise are not reported separately. Marble, breccia, and onyx are combined in the statistical reports, and as marble constitutes by far the largest proportion of the tonnage imported it is impossible to determine the amount from any country except Mexico. Recent marble imports from that country, all of which may be regarded as onyx marble, are shown in the following table.

Onyx marble (rough and manufactures) imported into the United States from Mexico. 1920-31

Year	Rough onyx		Manu- factures, value	Total value
	Cubic feet	Value		
1920.....	7,846	\$43,878	\$173	\$44,051
1921.....	2,245	9,191	137	9,328
1922.....	6,550	37,020	5	37,025
1923.....	5,349	24,933	24,933
1924.....	5,506	19,917	602	20,519
1925.....	2,767	16,163	16,163
1926.....	16,270	72,388	383	72,772
1927.....	26,509	96,062	2,264	98,326
1928.....	24,631	97,855	114	97,969
1929.....	18,687	76,889	1,277	78,166
1930.....	17,203	69,120	724	69,844
1931.....	1,475	12,000	69	12,069

The Tariff Act of 1930 provides duties on onyx and manufactures of onyx imported for consumption in the United States under paragraph 232, as follows:

(a) Marble, breccia, and onyx, in block, rough or squared, only, 65 cents per cubic foot; marble, breccia, and onyx, sawed or dressed, over two inches in thickness, \$1 per cubic foot.

(b) Slabs and paving tiles of marble, breccia, or onyx; containing not less than four superficial inches, if not more than 1 inch in thickness, 8 cents per superficial foot; if more than 1 inch and not more than $1\frac{1}{2}$ inches in thickness, 10 cents per superficial foot; if more than $1\frac{1}{2}$ inches and not more than 2 inches in thickness, 13 cents per superficial foot; in addition thereto on all the foregoing, if rubbed in whole or in part, 3 cents per superficial foot, or if polished in part (whether or not rubbed), 6 cents per superficial foot.

(c) Mosaic cubes of marble, breccia, or onyx, not exceeding 2 cubic inches in size, if loose, one-fourth of 1 cent per pound and 20 per centum ad valorem; if attached to paper or other material, 5 cents per superficial foot and 35 per centum ad valorem.

(d) Marble, breccia, and onyx, wholly or partly manufactured into monuments, benches, vases, and other articles, and articles of which these substances or any of them is the component material of chief value, not specially provided for, 50 per centum ad valorem.

Prices

As onyx is ornamental in character and quite variable in quality there are no standard methods of grading and no regular market quotations. The price varies with the quality and with the size of solid blocks. An onyx that might be worth \$5 a cubic foot in small pieces might be worth \$25 or \$30 a cubic foot if available in large solid masses.

TRAVERTINE

Occurrence

Travertine deposits large enough to supply blocks of commercial size are uncommon. Italy has been the chief source of production and has supplied the entire requirements of the United States until the last few years, during which time several deposits in this country have been developed. The Italian and domestic deposits are described briefly.

Italy.--Travertine appears to have been used in Italy as early as the second century B.C. and was employed at first only for ornamental purposes. It was later employed widely as a building stone.

The principal source of supply of travertine has been from the deposits at Tivoli about 16 miles east of Rome. Del Barco near the famous baths of

Acque Albule at Bagni, a railway station between Rome and Tivoli, is one of the oldest quarries. It furnished travertine during the days of the Roman Empire.

For several centuries the method of block separation at these quarries was by a slow process of cutting hand-picked channels on four sides and then slogging up at the floor. The stone is now cut with wire saws into blocks about 10 meters long, 1 meter thick, and 2 to 3 meters high. It has been reported that blocks can be put on cars at Bagni at a cost of about 350 lire a cubic meter and on board trans-Atlantic ships for about 400 lire. In 1929, the time of this estimate, 19 lire were equivalent to about \$1. The above costs, therefore, were respectively about 56 and 71 cents a cubic foot. Rock at the western end of the quarry is variegated gray and white, stained in places with iron oxide. During recent years this type has become more popular, whereas formerly the only kind used was the more regularly colored rock at the eastern end of the quarry.

The available travertine area is extensive, and several quarries other than the one described are worked. St Peter's, the greatest cathedral in the world, and the famous Colosseum, the largest theater, begun by Vespasian in A.D. 75 and dedicated by Titus in A.D. 80, were built chiefly of travertine from the Tivoli quarries. During the fifteenth and sixteenth centuries the Colosseum was treated as a quarry where stone was procured for many churches and palaces in Italy, notably the Palazzo di San Marco in Venice and the historic Palazzo Farnese in Rome. Demolition was finally stopped, and the structure partly restored.

During recent years large quantities of Italian travertine have been imported for interior decorative and structural uses in America. The Pisani quarry, not far from the Del Barco, supplied travertine for the Pennsylvania Railway Station in New York, a structure which attracted much attention and popularized the use of Italian travertine in this country.

California.--An old deposit in California, which had formerly furnished what was regarded as marble for buildings in San Francisco, was again operated in 1929. The quarry is located near the town of Bridgeport in Mono County. The stone comes in a variety of colors; some of it varying from clear white to pale yellow and gray is said to be of the same texture and quality as the best Roman travertine. There are also orange, pink, red, and brown colors.

Colorado.--A deposit that compares favorably with the famous Italian travertine has been developed about 6 miles east of Salida, Colo., near the Denver & Rio Grande Western Railway. The deposit forms one side of a mountain which rises 250 feet from the valley floor. The exposure is 1,300 feet long and 200 feet thick. The stone is light buff in color, very attractive in appearance, and is finding wide use in the cities of this country for both interior and exterior building.

Two other travertine quarries have been opened recently in Colorado, one near Salida in Chaffee County and one near Canon City in Fremont County. Production up to 1931 was confined to terrazzo.

Florida.--A deposit described as travertine occurs on the west coast of Florida in Manatee County near Bradenton. The deposit in which marketable material occurs covers an area of approximately 80 acres and extends to a depth of 100 feet in places. The stone exhibits the characteristic porous structure of Italian travertine and resembles it in general appearance. Considerable rock of high grade has been produced since 1929 and sold under the name "Floridene stone."

Georgia.--A quarry operated for a brief period near Cuthbert in southwestern Georgia provided stone of a brown to golden color and a porous structure. It was sold in limited quantities for interior building purposes, and waste material was marketed as chips for terrazzo floors. The product was classed commercially as travertine.

Montana.--Production of travertine began in 1932 at Gardiner, Mont., near the north entrance to Yellowstone Park.

Other States.--Certain porous beds in limestone deposits near Winona, Minn., Bedford and Bloomington, Ind., and Cedar Park, Tex., are sold as travertine.

Uses

The adaptability of travertine for exterior walls has been proved by its durability in the old Roman buildings. Domestic travertine, particularly that from Colorado, is also used for exteriors, but both the domestic and imported products are employed principally for interior building as flooring and wainscoting or for the walls of lobbies and entrances in large structures such as railway stations, hotels, banks, or other public buildings. The soft luster of travertine gives it decided warmth and renders it especially desirable for such uses. Because of its lightness and strength it is favored for arches and other vaulted structures. It is claimed that travertine improves the acoustics of large rooms or auditoriums. It is especially desirable for floors because of its nonslipping quality and nonfatiguing effect upon pedestrians, which are attributed to the holes or voids in the stone. It resists abrasion remarkably well.

In the construction of homes it is employed for fireplaces, doorways, lintels, flagging, and floor tile. Small pieces are in demand for table tops, lamp bases, and novelties. Waste material may be used for manufacture into mosaic tile, terrazzo, and stucco.

Notable examples of the use of travertine are in the interior of the Pennsylvania Station in New York and in the United States Chamber of Commerce Building and the Department of Commerce Building in Washington. In the latter it is employed for floors of the entrances and main lobby and for the floors and treads of the stairways leading from the main lobby to the basement and to the second and third floors.

Production and Imports

Production figures of travertine are included with those of limestone in the statistical reports of the Bureau and therefore cannot be shown separately. An increase in the production units of travertine would indicate a growing demand for this product.

Travertine was not separated from other stone imports prior to 1930. It has been brought out in tariff hearings that imports normally amount to about 100,000 cubic feet a year and that the material sells in New York at about \$2.25 a cubic foot. The chief markets are in the large Eastern Cities.

INFORMATION CIRCULAR
UNITED STATES BUREAU OF MINES

EXPLOSIONS IN UTAH COAL MINES,
1900 - 1932¹

By D. J. Parker²

INTRODUCTION

In attempting to discuss a subject of this character it should be clearly understood at the outset that the purpose is not to emphasize the mere fact of occurrence of such catastrophes, the memory of which time has largely relegated to oblivion, but rather to stress the means of their prevention, available not only subsequent to but also at the time of occurrence.

A careful, conscientious, unbiased study of the causes of mine explosions leads inevitably and definitely to the conclusion that the human element is the most important factor. The human factor should however, be evaluated in terms of responsibility--responsibility on the part of the employer to recognize the existing hazards and take the necessary remedial action, and on the part of the employee to perform his duties faithfully, diligently, scrupulously, and in strict conformity with State mining laws and company rules or other regulations.

While it is not the purpose of this report to endeavor to establish individual responsibility for any particular disaster it is important that the management accept the responsibility in nearly all instances for the prevention of such accidents.

The several causes of mine explosions in Utah will be discussed and analyzed, and definite recommendations will be made based on obvious conclusions drawn from the experience of the coal-mining industry, with a view to revitalizing the interest and concern prerequisite to the prevention of accidents as utterly unnecessary as mine explosions. The progress heretofore made in greater safety has been based largely on sad and costly experience, regardless of the type of industry involved.

1 The Bureau will welcome reprinting of this paper, provided the following footnote acknowledgment is used: "Reprinted from U. S. Bureau of Mines Information Circular 6752."

2 District engineer, U. S. Bureau of Mines Safety Station, Salt Lake City, Utah.

HISTORICAL REVIEW³

The records of early exploration of Utah start with two Spanish priests--Escalante and Domingues, of Sandoval, N. Mex.--who reached the shores of what is now known as Utah Lake on September 23, 1776. Forty-eight years later, in 1824, Great Salt Lake was discovered by James Bridger, a trapper who came down the Bear River, and in 1832 a party under Captain Bonneville explored Salt Lake Valley.

If these early explorers discovered any coal, no record of the discovery has been preserved. On July 24, 1847, the first party of Mormon immigrants, led by Brigham Young, entered Salt Lake Valley and established the first permanent settlement. How soon after their arrival coal was discovered is difficult to ascertain. Marcus E. Jones, Treasury expert, reported to the Treasurer of the United States in 1890 that the first discovery of coal was made in Sanpete Valley and that in 1851 coal was discovered near Cedar City in the Colob-Kanab field.

Bulletin 285 of the United States Geological Survey says that coal was discovered at Wales, Sanpete County, in 1857 and mined in 1862; the 1896 report of the coal-mine inspector of Utah says that the Wales mine was the oldest coal mine in the State, having been opened in 1855, when the output was consumed by the surrounding settlements. It seems definitely established, therefore, that the Wales mine has the distinction of being the first worked. It is also known that the Winterquarters mine, opened in 1878, was the first in Carbon County. The Union Pacific Railroad was completed across Utah in 1869 and connected with Salt Lake by a branch line in 1870. The Wales mine was reached by rail in the early seventies. When Winterquarters was opened a narrow-gage railroad was accessible for shipping the coal to Provo.

FATALITIES

Table 1 lists 12 explosions, only 9 of which resulted in loss of life. The second and most disastrous explosion, also one of the worst in the mining history of the United States, with a loss of 200 lives, occurred May 1, 1900. The next in rank with respect to the number of lives involved (referred to as case 4) occurred on March 8, 1924, with a loss of 171 lives. Case 11 was the third highest, with a loss of 20 lives; cases 6 and 12 were next, with a loss of 5 lives each; cases 3, 5, and 10 accounted for two lives each; and case 8 one life. In cases 1, 7, and 9 no lives were lost because there were no men in the mine at the time.

Of the 9 explosions resulting in loss of life, 5 were major disasters, that is, those in which 5 or more persons were killed.

The total number of lives lost due to explosions in Utah from May 1, 1900 to July 1, 1933 was 408, and 90 percent of this total is due to the two largest explosions listed in table 1 as cases 2 and 4. From July 1, 1918 to June 30, 1933, inclusive, a period of 15 years, there were 528 fatalities in the coal mines of Utah from all causes, and 39 percent were killed by explosions.

³ Allen, C.A., Tech. Paper 345, 1925, 90 pp. Analyses of Utah Coals.

From 1900 to 1932, inclusive, 110,282,580 tons of coal were produced; the number of tons produced per fatality from explosions alone was 270,300, and the number of fatalities from this cause per million tons mined was 3.6. Hence for this period the fatality rate from coal-mine explosions per million tons of production in Utah is higher than the average fatality rate of the coal mines of the United States for all causes of coal-mine accidents (falls, haulage, explosions, explosives, electricity, etc.).

Following the Winterquarters disaster on May 1, 1900, with its 200 fatalities, there was not a single major disaster due to explosions for almost 24 years. This is a rather remarkable record when it is considered that during this time practically no rock-dust was used, black powder was a common blasting agent in many of the mines for a number of years, blasting was done during the working shift in several mines, water was not very generously applied to cutter bars or roadways and face regions, nonpermissible electrical equipment was in more or less general use, and fireproof or fire-resistant stoppings were installed to only a limited extent. The immunity from explosions over at least the last 10 years of this period may be attributed largely to good fortune.

Since explosions do not happen by chance but from definite causes their prevention would likewise appear to be the result of well-directed effort; however, the rigid precautionary measures that were in effect for a number of years following the Winterquarters explosion tended to lull the industry into a false sense of security, and apparent relaxation of many safety practices and methods set in about the period of the World War.

CAUSES OF EXPLOSIONS

It is not always a simple matter to determine the exact cause--that is, the igniting agent--of an explosion; furthermore, frequently there are differences of opinion among mining men as to causation influences or agencies as found in investigations after such catastrophes. Some reasons for this are that those nearest the origin are often killed, the forces of the explosion tend to destroy the evidence, and not infrequently the initial explosion may be followed by one or more secondary explosions locally or even extending throughout the mine, thereby confusing the evidence of the cause of the first.

The data in table 1 on causes are based on such records as are available. The evidence, however, is not always conclusive; as a result, in some cases there may be divergency of opinion. It will be noted that 3 explosions, 2 covering the entire mine and 1 about half of the mine, resulted in neither injury nor fatality to anyone.

Of the 9 explosions resulting in 1 or more fatalities, 3 were caused by open lights, with 175 or 43.2 percent killed; 1 was caused by explosives (black blasting powder), with 200 or 49.4 percent killed; and 5 were caused by electricity, with 30 or 7.4 percent killed.

Attention might be directed to the fact that 5 of the 12 explosions occurred in 2 mines, 2 in 1 mine and 3 in the other. Cases 4 and 7 refer to the same mine, and the other mine is designated by cases 5, 8, and 9. (See table 1.)

It would appear advantageous to group the 12 explosions by causes--that is, open lights, explosives and blasting, and electricity--and discuss briefly each case falling in its respective group.

OPEN LIGHTS

There is no known record of an explosion in Utah mines ever having been caused by matches or smoking material. Since the beginning of Statewide use of closed lights (permissible electric cap lamps) in mines of this State about the middle of 1924, no explosion or gas or dust ignition has been initiated by an open light, an improperly assembled flame safety lamp, or a defective permissible electric lamp.

The first five explosions of which there is record, covering the period from March 22, 1900, to May 23, 1924, were all in open-light mines, although permissible electric cap lamps were at hand at one mine but not yet installed when the explosion listed as case 4 occurred. These five explosions killed 375 men (92.6 percent of the total fatalities from explosions since 1900). Thirty men (7.4 percent of the total) have been killed by explosions since closed lights were introduced and in no instance was the closed light even remotely responsible for the ignition.

Case 3.--At the noon hour two men building a stopping proceeded into the air course, where it was warmer, to eat their lunch. A body of gas which had collected in a cavity in the roof was ignited by their open lights, and both men were fatally burned. The concussion was sufficient to blow out timbers, with resultant roof fall, which contributed further to the injury of the two men. This accident indicates poor ventilation and lack of proper gas inspection.

Case 4.--This explosion, which resulted in the death of 171 men (every man in the mine), was the second largest with respect to loss of life that has ever occurred in Utah; in fact, only 10 explosions have caused greater loss of life in coal mines of the United States.

There appeared to be enough evidence to establish the fact that this explosion originated in a room driven to the rise from an entry, the pitch of the coal bed being about 10 percent and thickness about 25 feet. The night before the explosion some of the top coal had been shot down in this room to within a few feet of the face. The fire boss was found on the inby side of this pile of top coal with his flame safety lamp, which had been disassembled. Two miners were found on the outby side of the coal pile close to a car of coal, which evidently had been loaded that morning. There is every indication that the fire boss had found gas near the roof in this room on his initial examination before the shift came into the mine and had gone back with the two miners to remove it--anything but safe procedure,

TABLE 1.—Gas and dust explosions in Utah coal mines, 1900-32

Case	Date	Mine	Type of mine	Tons per day	Possible means of prevention at time	Explosion			Causes of gas accumulation	Cause of ignition	Other factors	Men in mine		Killed	Injured	Escaped unassisted
						Type	Extent	Limited by				Average	At time of explosion			
1	Mar. 22, 1900	No. 1	Slope; open lights.	1,100	More care in blasting	Dust	Entire mine.	Not limited	-----	Blasting	Faulty detail in blasting practice.	325	0	0	0	0
2	May 1, 1900	Nos. 1 and 4	Drift; open lights.	---	Removal of dust, use of water, proper handling of explosives.	do	do	do	Explosive gas was never found in this mine.	Probably blasting.	Blasting with shift in mine.	312	312	200	7	No. 4 - 2 No. 1 - 103
3	Mar. 3, 1923	No. 3	Slope; open lights.	2,000	Proper gas inspection and supervision, closed lights.	Gas	Local	Probably dampness.	Cavity in roof.	Open light.	Faulty inspection and ventilation.	350	350	2	0	0
4	Mar. 8, 1924	No. 2	Drift; slope; open lights.	1,400	Rock-dust, water, good ventilation, and closed lights.	Gas and dust.	Entire mine.	Not limited	Faulty system of mining top coal.	Open light or flame safety lamp	Poor supervision, gas inspection, and ventilation.	171	171	171	0	0
5	May 23, 1924	No. 1	Slope; open lights.	2,000	Efficient ventilation, better supervision, closed lights.	Gas	Local	Wetted area	Poor ventilation	Open light.	Brushing gas with water from hose.	300	300	2	0	0
6	Sept. 21, 1924	do	Slope; closed lights.	1,000	Rock-dust, water, and good ventilation; permissible electrical equipment.	Gas and dust.	Entire mine.	Not limited	Lack of ventilation.	Electric mining machine	Fan had been shut down for repairs.	175	5	5	0	0
7	Aug. 8, 1928	No. 2	Drift; slope; closed lights.	2,500	Insufficient rock-dust, proper use of explosives.	Dust and probably gas	do	do	-----	Blasting	Faulty shooting practice.	130	0	0	0	0
8	Jan. 24, 1929	No. 1	Slope; closed lights.	2,000	Permissible locomotive, improved ventilation.	Gas	Local	Rock-dust and water.	Poor ventilation	Arc from locomotive controller.	Cavity in roof.	150	150	1	0	0
9	Jan. 25, 1929	do	do	2,000	Increased rock-dusting.	Dust and probably gas.	One half of mine.	Rock-dust barriers and water.	-----	Explosives	Probably poorly placed shot.	150	0	0	0	0
10	Jan. 20, 1930	No. 3	do	750	Permissible electrical equipment.	Gas	Local	Wet conditions.	Outrush of gas	Electric mining machine.	Nonpermissible mining machine.	100	Unknown	2	0	0
11	Feb. 6, 1930	Nos. 1 and 3.	Tunnel; slopes; closed lights.	1,100 600	Proper gas inspection, rock-dust, permissible electrical equipment.	Dust and gas.	Local in No. 1 violent in No. 3	Wet conditions in No. 1 and expansion in No. 3	Probably released by mining machine.	Sparks from machine bits or electric sparks or electric mining machine.	Too infrequent gas inspection	166	26	No. 1 - 17 No. 3 - 3	2	7
12	Mar. 8, 1930	New	Slope; closed lights.	150	Good ventilation and supervision, permissible electrical equip.	Gas.	Local, little violence.	Wet conditions.	Fan down, insufficient natural ventilation.	Electric mining machine.	Poor supervision.	50	13	5	6	2
Total	--	--	----	--	-----	----	----	----	-----	----	-----	--	----	408	15	114

especially in an open-light mine. Evidently he had climbed on top of the coal pile which had been shot down the night before, and while he was testing for gas in the high place from which the top coal had fallen the flame on his flame safety lamp was extinguished by an explosive mixture of gas and air. After getting down from the pile of coal and apparently having used a spray of water from a hose to diffuse and partly remove from the high place the explosive gas accumulation the fire boss opened his flame safety lamp and was in the act of relighting it, using his carbide lamp, when he or one of the miners ignited the gas. The gas very probably was stirred up by the fire boss while making his test and while using the water spray. A roof fall, which occurred just before or at the time of the explosion and partly covered one of the miners, may have pulled the gas down to a point where it could easily have been ignited by one of the miners' carbide lamps; more probably, use of the water spray moved the gas so that it came in contact with an open light.

The first explosion apparently was initiated by a local gas ignition of sufficient magnitude to cause it to spread to adjacent places where there was much dust and possibly additional gas, and this precipitated a widespread, extremely violent, dust explosion. A few minutes after the first explosion a second occurred in which gas presumably played little or no part. While it was thought by many that a third explosion occurred, this was not substantiated by the evidence.

No rock dust had been used in this mine before the explosion, water being relied upon to render the coal dust nonexplosive. This occurrence emphasizes the fallacy of relying upon the use of water only as an agent to make Utah coal dust safe. Assuming that the unstinted use of water might serve to prevent fine coal from participating in an explosion, due to the human element involved, there would always be the probability of sufficient laxity to destroy in a large measure the reliability of this method of treating coal dust.

Permissible electric cap lamps had been delivered to the mine before the explosion but had not been put into service, pending arrival of part of the charging equipment; although the explosion might not have occurred if the electric cap lamps were being worn, on the other hand there is no guaranty that the fire boss might not have used a match to relight his flame safety lamp and thereby precipitated the disaster, as there was every evidence that he was attempting to relight the flame safety lamp when killed by the explosion.

This mine was one of the most safely equipped and operated in Utah; in fact, it was considered by many persons as a model mine with respect to safety. More than 180,000 cubic feet of air entered the mine every minute, and ventilation was by splits. The mine was considered one of the best-sprinkled properties in the State; sprinkling hose with adequate water pressure were maintained at each working face for use of those employed at or near the faces, and three full-time sprinklers were regularly employed to keep haulage roads and nonworking places dampened. However, certain parts of the mine, including that in which the explosion started, were very gassy, the coal dust is

exceptionally explosive, the coal is of such nature as to give off much dust, the high coal worked on a pitch made effective ventilation difficult, the open lights and nonpermissible flame safety lamps were dangerous, and the procedure in removing an accumulation of explosive gas was anything but safe.

Case 5.--Two miners were fatally burned when they attempted to remove standing gas in their working place by "brushing" it out with water from hose, and the gas was ignited by naked lights in use in the mine. The place was "danger-marked", and the information is to the effect that the two miners, who were of foreign birth, had been warned to stay out of the place. It is possible that they did not fully understand the orders, as was brought out by some conflicting evidence. The explosion was local and was undoubtedly limited by the presence of moisture.

Substantially erected line canvas from the last open crosscut close to the face, well-maintained with a minimum of leakage, will under ordinary conditions keep the face free from gas accumulations. The proper time for removing standing gas is before the shift enters the mine, and then its removal should be attempted only by properly qualified persons taking all feasible precautions.

EXPLOSIVES AND BLASTING

Although there have been at least four explosions, presumably caused by explosives or blasting, only one resulted in loss of life. Since the introduction of permissible explosives, fired electrically, in Utah no mine explosion which resulted in a single fatality has been caused by blasting. Three explosions occurred at the time of blasting electrically from the surface with no one underground and no person was hurt on the surface or elsewhere; in 1 instance dynamite was used, and in 2 instances permissible explosives were employed. It is probable that in at least two cases the explosion was due to ignition of gas or dust or both by an electric arc rather than by the explosive.

Case 1.--When the explosion occurred this mine was one of the most progressive in the United States from a safety point of view; in his report to the Governor of Utah for 1899, the State mine inspector said of this property: "To guard against dust danger, the management has put in water and steam sprays to keep the mine damp."

"The coal is dug in the mine by being first undermined and shot off with Hercules powder, of which they used 38,450 pounds during the year 1899. All shots are fired by electricity, when all men are out of the mine, which I think is the safest way to mine coal where dust is explosive."

At about 7:30 on March 22, 1900, a shotfirer threw in the final shooting switch on the surface to put off all of the shots in the mine. He stated that the resultant concussion did not appear to be as loud or as strong as usual, so fearing that some shots had not "gone", he threw the electrical current into the shooting lines a second time. Immediately a very violent explosion occurred which blew out every masonry stopping and overcast in the mine, practically demolished about 200 mine cars, and in general wrecked the interior of the mine.

The State mine inspector's report to the Governor on this explosion said: "The explosion was nothing more than a dust explosion caused by the firing of heavy shots on the ninth rise. ***** When the shots went off, they caused a great dust, and it is supposed caused some of the wires to become crossed, and when the current was turned on the second time, it caused a big flash and ignited the dust and caused the explosion."

While the property damage was heavy, no human being suffered any injury, an excellent tribute to the foresight of the mining company which before 1900 had in operation a system of blasting which was and is thoroughly effective in safeguarding human life in a mine known to have explosive gas and very inflammable dust.

Case 2.--According to the best available information, this explosion, which cost the lives of 200 men, originated in No. 1 mine but both No. 1 and No. 4 were involved. The explosion occurred about 10:25 A.M. on May 1, 1900. Seven were injured, 2 escaped from No. 4 mine and 103 from No. 1 mine, and about 84 employees of No. 4 mine and 116 from No. 1 mine were killed, the latter chiefly by gases.

The cause has never been absolutely determined. The theory has been advanced that firing shots in rotation raised a cloud of dust of sufficient intensity to permit ignition by subsequent shots, but the most probable theory appears to be that a quantity of black blasting powder was accidentally ignited by an open light, initiating the explosion, which was propagated by coal dust. Explosive gas is not known ever to have been found in either No. 1 or No. 4 mine from the time they were opened until they were finally worked out and abandoned, and there is absolutely no question that this was entirely a dust explosion.

When it is considered that mines at that time were operating with open lights, using black blasting powder, in many instances with several kegs in one working place, with shots stemmed with fine coal, and few if any limitations as to quantity of explosive per hole and firing done by miners with the shift in the mine, with absence of rock dust and little or no use of water to neutralize the coal dust, it is marvelous that there were not more disasters of this character.

The explosion in these two mines exemplified not only the hazard of open lights and black blasting powder in coal mines but was one of the most convincing instances of the explosibility of coal dust without the aid of explosive gas that had ever occurred and in fact that has occurred to this day. Moreover, it pointed out the hazard of connecting coal mines; in this instance, the explosion started in one mine, killed about 84 persons, and wrecked the mine; while the other mine was essentially unharmed by the explosion, the poisonous gases extended into the other mine and killed over 100 employees.

Case 7.--This explosion was essentially as violent as the one in this mine on March 8, 1924, although fortunately no lives were lost. The effects of the explosion extended to every part of the mine, wrecking concrete overcasts and blowing out nearly all the stoppings, which were also concrete.

Several fires resulted, which necessitated sealing a portion of the mine, and a local explosion occurred in the sealed area about the time the last seal was being completed, but with no damage to the seals and without injury to any workers.

The explosion occurred immediately after the closing of the master shot-firing switch on the surface. No one was in the mine at the time, and no one was injured on the surface.

There was much evidence of overcharged shot holes, and it is probable that explosive gas may have been present in one or more working faces. It appears to have been the custom to drill some shot holes to a depth of as much as 18 feet, slightly inclined upward to bring down top coal, and numerous shots of this type were fired in the same working places at one time. In view of such obviously unsafe practices in vogue before the explosion the occurrence was not wholly unanticipated by numerous well-informed mining men familiar with the conditions.

Although numerous questionable practices undoubtedly were employed with respect to blasting in this mine before this particular explosion, it is to the credit of the management that the system of blasting electrically from the surface with no one in the mine, which was adopted in this coal-mining community many years before, was in effect at the time of this explosion; therefore while the property loss was very heavy, no person was killed or injured.

Case 9.--This case also definitely proved the excellent safety policy of firing all shots from the surface with no one in the mine. The explosion occurred simultaneously with the closing of the master shot-firing switch on the surface. No one was injured, but considerable damage was done to that part of the mine traversed by the explosion wave. Concrete stoppings and overcasts were blown out; and many timbers were dislodged, followed by several heavy roof falls.

This mine is recognized as one of the most gassy in the State, and the coal dust is highly explosive. There was, however, no evidence of violence more than 3,000 feet in any direction from the origin of the explosion, and only about half the mine was affected. This affords one of the clearest-cut instances on record where a violent explosion was limited by rock-dust and rock-dust barriers plus wet zones.

The explosion originated in the face of an entry less than 200 feet long. A fire was started in this entry as a result of the explosion and was extinguished by local flooding. After the fire had been extinguished the entry was found to be caved to such an extent that it was never reopened. In consequence, evidence on the underlying cause of the explosion that might have been available at the face was lost.

The explosion may have been started by a misplaced shot, or gas may have been present as the result of a possible fall which may have brought down the

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line brattice between the time the shot holes were charged and the firing of the shots. Incidentally, this is one of the best-supervised mines in the State, as evidenced by operating the up-to-date shot-firing system, the unremitting effort to maintain good ventilation, and numerous other practices and installations in line with safe mining practice in a property with numerous inherent hazards.

ELECTRICITY

Although electricity has become a most important economic factor in the production of coal, on the other hand its wide, varied, and extensive application has created one of the greatest hazards in modern coal mining.

Electricity was either directly or indirectly responsible for 5 of the 12 explosions. Four of the five were fairly clean-cut cases, but in the fifth there was the possibility that gas was ignited from sparks caused by mining-machine bits cutting against a rock spar, not an uncommon occurrence in that section of the mine.

Electricity accounts for starting 45 percent of the explosions that have occurred in Utah. Thirty-three fatalities (8.0 percent of the total number (408) killed by explosions in the coal mines of Utah) were due to ignitions by electricity.

It will be noted from table 1 that 4 of the 5 explosions of electrical origin were caused by electric coal-cutting machines (nonpermissible type) and the other by an electric (trolley-reel type) locomotive.

Case 6.--This explosion occurred about 6:00 p.m., with only five men in the mine, all of whom were killed. A few minutes later a second explosion occurred, the cause of which was never satisfactorily determined. The initial explosion was comparatively light, while the second explosion was violent, covering all parts of the mine. As an indication of the high pressures developed, a 5- by 5-foot timbered intake-air shaft, 50 feet deep, was increased to about 8 by 10 feet in cross section, all of the timber and loose rock between the timber and the solid rock being scattered along the surface adjacent to the shaft collar.

This was a closed-light mine, permissible electric cap lamps being used by the underground force. Magnetically locked permissible flame safety lamps were used for gas testing.

The mine had been partly rock-dusted which, as in other instances, may have given a false sense of security.

The evidence indicated fairly conclusively that a body of standing gas was ignited by sparks or arcs from a mining machine. It was stated that on the day of the explosion a curtain had been placed across the slope, which was one of the two intakes, to deflect the air into a section of the mine which was being rock-dusted. Assuming this to be true, that section of the

mine in which the explosion is supposed to have originated would be deprived, for the time being, of the proper amount of air. It was further stated that at the time of the explosion the fan had been shut down for emergency repairs.

This mine was classed as gassy, and either of the above incidents might account for the presence of standing gas.

Case 8.--A motorman was fatally burned by a local gas explosion while pushing four empty cars into a slant or inclined crosscut with a trolley-reel locomotive operating from the trailing cable. The locomotive had reached a knuckle in the slant crosscut, and a roof cavity over this point was apparently filled with gas. The movement of the cars probably brought the gas down to a level with the controller.

Proper inspection and subsequent removal of the gas, or the use of a permissible storage-battery locomotive, would have prevented the accident. A dust explosion, possibly extending throughout the mine, was prevented by rock-dust and damp to wet conditions at the point of origin.

Case 10.--A machine runner and his helper, after cutting a place, had loaded the machine on the truck when a local gas explosion caused their death. This place had been inspected by the fire boss $3\frac{1}{2}$ hours before it was cut and found clear of gas. The line brattice extended to within $6\frac{1}{2}$ feet of the face. The nonpermissible machine had evidently cut into a "blower" which filled the face region with gas. The gas was ignited by an electric arc or spark or from an overheated resistance. The explosion, which was of sufficient violence to blow the nips of the trailing cable off the power circuit, was limited by the naturally wet conditions which extended 120 feet back from the face.

Case 11.--The explosion originated in No. 3 mine and extended through a rock tunnel, 2400 feet in length, into No. 1 mine. Three men were in No. 3 mine and 26 in No. 1 mine; this constituted the entire night shift of both mines. All in No. 3 mine were killed and 17 in No. 1 mine.

In No. 3 the explosion was violent and covered virtually the entire mine. Little violence was manifested in No. 1 mine, only two of the 17 being killed by violence. The rock tunnel connecting the two mines was naturally wet to such an extent that no flame reached No. 1 mine; 15 of the 17 perished from the effects of the afterdamp in this mine. Both mines had been partly rock-dusted; this again illustrates the futility of relying on "half-way" explosion-prevention measures.

The last inspection for gas in the place in which the explosion is supposed to have originated was made about 15 hours before the machinemen began cutting the place. There were two mining machines, both of nonpermissible type, in operation at the time of the explosion. One was traveling outby in a room, while the other was undercutting in another section of the mine.

A body of gas was ignited by one of the two machines or possibly by a spark caused by machine bits cutting a rock spar. The gas may have been released by the undercutting. There is also a possibility that 1 of 3 doors may have been left open, thereby short-circuiting the air.

Case 12.--Of the 13 men in the mine at the time of the explosion, 5 were killed, 6 were injured, and 2 escaped uninjured.

The mine was in the initial stages of development and was recognized as extremely gassy. In spite of this fact the fan was not in operation at the time of the explosion, neither was line brattice used to ventilate the faces, 3/4-inch compressed air lines being relied upon to give the necessary face ventilation.

The evidence, while perhaps not absolutely conclusive, indicated beyond a reasonable doubt that the explosion was caused by ignition of gas by a non-permissible electric mining machine. The evidence further indicated that this was a gas explosion in which coal dust played a minor part, due to the limited extent of the mine workings and the liberal use of water on cutter bars and the face regions. No rock-dusting had been done up to the time of the explosion. The mine was on a strictly closed-light basis, but the non-smoking rule was enforced with laxity if at all.

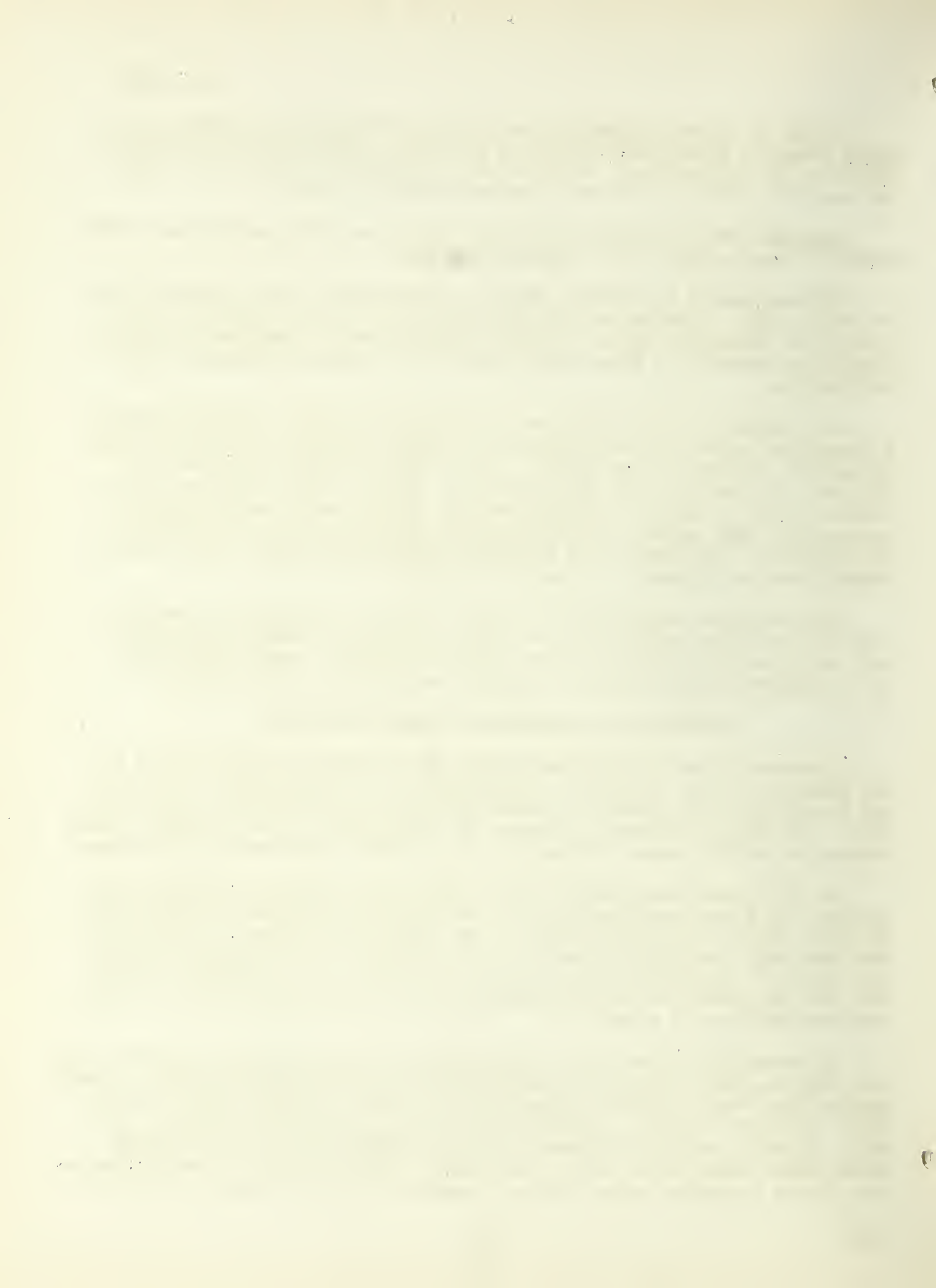
This explosion illustrates the folly of trying to operate a gassy mine without enough face ventilation and with nonpermissible electrical equipment and is another instance of the danger of trying to move explosive gas or to ventilate gassy regions by a jet of compressed air or of water.

PREVENTION OF EXPLOSIONS IN UTAH COAL MINES

A review of the foregoing occurrences is of value only in so far as it may impress upon the industry the possibility of the recurrence of explosions and to emphasize the lessons to be learned from those which have already happened. Of greater importance, however, is application of the knowledge gained through scientific research relative to the methods of preventing explosions.

Of all accidents occurring in coal mines there appears to be least justification for those resulting from mine explosions. The United States Bureau of Mines, through large-scale tests at its Experimental mine, has proved that dust from Utah coals ranks among the most explosive in the country. Through such tests it was determined that at least 70 percent of incombustible material must be present to prevent propagation of an explosion by Utah coal dust when explosive gas is absent.

Rock-dusting; rock-dust barriers; water applied generously to cutter bars, face regions, coal piles, empty and loaded cars, and haulage roadways; adequate ventilation; rigid fire-boss inspection, with accurate daily reports; permissible explosives, used in a permissible manner; permissible electrical equipment, properly maintained; closed lights, of permissible type; permissible flame safety lamps; tight cars; clean tracks; limited topping of cars; enforcement of the no-smoking rule, plus good supervision--all of which are within



the reach of the average mine operator--will go far toward preventing dust explosions and reduce the likelihood of gas ignitions to a remote possibility.

ROCK-DUST

Experiments in the use of incombustible dust or rock-dust to reduce the coal-dust hazard in mines were conducted in Great Britain and France as far back as 1906 and 1907, and the United States Bureau of Mines determined in 1913 the effectiveness of rock-dusting and so stated in its publications.

It required many years of intensive educational work to convince the bituminous-coal-mining industry that coal dust in the absence of inflammable gas is explosive. For more than 20 years the Bureau of Mines has endeavored to gain general recognition of the fact that rock-dust properly applied in connection with coal dust would prevent the propagation of coal-dust explosions, yet at this time not one percent of the coal mines of the United States are effectively protected by rock-dusting.

To prevent the propagation of mine explosions, the Bureau of Mines recommends⁴ rock-dusting in all coal mines, except anthracite mines, in every accessible part, whether in damp or dry condition. It also recommends that rock-dust barriers be used to sectionalize the mine as additional defense; but they should not be regarded as a substitute for generalized rock-dusting.

Rock-dusting methods are based upon the facts (1) that coal dust when mixed with sufficient incombustible dust will not explode, burn, or assist in propagating an explosion, and (2) that dry, fine, incombustible dust raised into the air in a sufficiently dense cloud will tend to check and stop a coal-dust explosion. These facts have been demonstrated by many experimental tests in this country and in Europe, and after both local and general explosions in operating mines.

Rock-dusting a mine is likely to be of little or no value in preventing the propagation of an explosion unless it is thoroughly done and well maintained. Rock-dusting insufficiently done leaves a false sense of security, if gas or dust is ignited and an explosion started, which continues along undusted aircourses or poorly dusted entries, emphasis is likely to be placed upon the "failure" of rock-dusting rather than upon the insufficient dusting.

The first step in the adequate rock-dusting of a mine should be thorough cleaning by washing down of all passageways and loading out of as much loose and gobbled material as practical. Rock-dusting should be started in the sections of a mine where explosions would be most likely to start, such as dry sections liberating considerable gas. Producing entries should generally be the first entries dusted. A mine should be

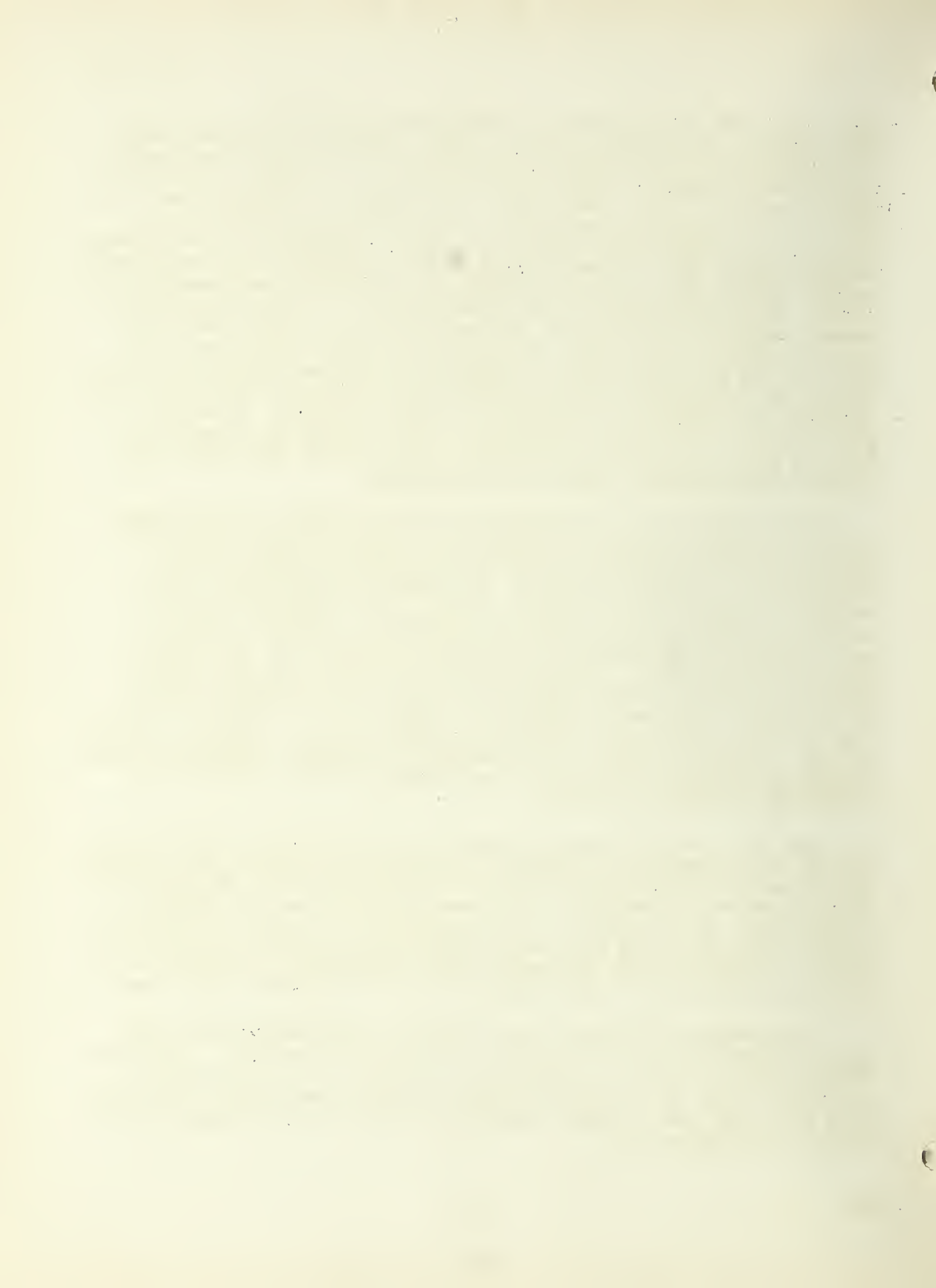
⁴ U. S. Bureau of Mines Accident Prevention Course, Section 2, Explosion and Fire Prevention

rock-dusted from the face outby, and rock-dusting advanced as the workings advance. All crosscuts, rooms, pillar workings, and entries should be dusted within at least 40 feet of the face or to the last breakthrough. A commendable practice, rapidly coming into use, is to keep the rock dust within three cuts of the working face. All entries or workings from which track is to be removed, should receive a heavy coating of rock dust on ribs, roof, and floor before track is removed. Aircourses and other places having no track may be dusted by means of tubing carried from a rock-dusting machine through doors or holes in stoppings and along the aircourse to be dusted. There are machines on the market capable of blowing dust through several hundred feet of tubing. Good rock-dusting may be done either by hand or machine. Hand dusting of aircourses has been done by men applying dust from bags hung on their shoulders and transporting sacks of dust through the aircourses by sled. Another method used is to blow dust through stoppings into aircourses at intervals of 200 to 400 feet. If there is a good velocity of air in such aircourses, fine dust will carry well; if there is little air traveling, the rock-dusting done in this manner will be poor except near the points of dust discharge.

The amount of dust to be applied to an entry depends on the character of the coal, the amount of incombustible matter already present in the mine road, and rib dust, as well as the practices likely to cause deposition of coal dust day by day. In medium and high-volatile coals enough rock dust should be added to the rib, roof, and floor surfaces so that the road dust and the rib and roof dust, considered separately, each contain over 65 percent (not less than 70 percent for Utah coals) incombustible material. There should be enough additional dust added so that there will be a safety factor, that is, so that redusting will not be immediately necessary. Usually 3 to 4 pounds of rock dust per linear foot of entry is the minimum to apply on first dusting. Redusting, particularly of haulage entries, may be necessary at frequent intervals if there is much coal spillage or if there is much coal dust thrown into the air current by rapidly moving trips.

The best practical means of determining whether rock-dusting is sufficient is by sampling the mine dust and testing the samples for incombustible content. This should be done after rock-dusting and at frequent and regular intervals thereafter. After some experience has been gained in rock-dusting, one can judge to a considerable extent by eye as to the more poorly dusted places and take samples accordingly, although samples collected at definite points determined without reference to condition of the dust will probably give a truer picture of dust conditions in the mine.

The quantity of coal dust which a silver quarter-dollar will hold is the minimum amount per cubic foot of air that will explode. It is believed that this fact is not generally appreciated and it is readily apparent that very few coal mine workings have fine dust content as low as this very small amount, hence practically all parts of coal mines have the hazard of explosive dust.



EFFECT OF WATER ON EXPLOSIBILITY OF COAL DUST

Wet dusts can not be classified with dry dusts because of the tendency of the particles to cling together, making the formation of a dust cloud more difficult. Experiments have shown⁵ that the effectiveness of water in preventing scattering of dust is the only thing that can justify its use; at the working face there is certainly ample place for it. On the other hand, the use of water to render coal dust nonexplosive in mine passageways has been a flat failure, and there is experimental evidence to show that it can not succeed with bituminous coals. Tests were made in 1925 of mixtures of four Utah coal dusts and water. It was found in the course of this investigation that 20 percent water was as much as these dusts (size 20 percent through 200-mesh) would retain; if more was added, it sank through the dust and drained away, confirming in general earlier similar tests on Pittsburgh bed coal dust. Mixtures of two of the coals containing 20 percent water propagated explosions under standard propagation test conditions. A similar mixture of the third coal was subjected to a gas explosion and also propagated flame throughout the test zone. These tests leave little doubt concerning the explosibility of wet bituminous coal dusts. It is true that such wet dusts are raised into the air from the floor only with considerable difficulty, but they are dislodged from higher surfaces or timbers with ease, and when formed in a cloud by any means, they ignite and propagate flame despite the presence of the water.

There have been a number of disastrous explosions in supposedly well-watered mines, and even where water has collected in a low place or "swamp" for distances of 100 to 200 feet, an explosion flame has jumped over it, and has even coked dust floating on top of the water.

CONCLUSIONS

Mine explosions can be prevented by applying practices and methods now readily understood and available. They will be prevented only when there is firm determination on the part of the management to do so.

The three primary causes of the explosions described in this paper were explosives and blasting, open lights, and electricity. Only one of the four explosions chargeable to explosives and blasting resulted in loss of life; the other three resulting in no loss of life, are notable instances of the advantages to be derived from blasting with no one underground. The one explosion in Utah's coal mines with blasting as the cause and the loss of life occurred in 1900 when black blasting powder, open lights and blasting during the shift were allowed; all of these are now outlawed and it is significant that in mines with the unusual inherent hazards such as afflict the coal mines of Utah not a life has been lost in a Utah coal mine through an explosion caused by blasting, since blasting during the working shift has been prohibited.

The almost universal adoption of the closed light in the coal mines of Utah was a tremendous step in the right direction. At least 175 persons have been killed in the coal mines of Utah through ignitions by open lights, but

5 U. S. Bureau of Mines Accident Prevention Course, Section 2, Explosion and Fire Prevention.

not a single life has been lost through the use of the permissible electric cap lamp. Some of the smaller or "wagon" mines are now using the open light, and sooner or later fatalities are almost certain to occur by fires, explosions, or other untoward occurrence brought about by open lights.

Four at least (if not all) of the explosions attributable to electricity probably would not have occurred had permissible equipment been in use at the time. Through the cooperation and foresight of the Industrial Commission and the Mine Inspection Department, permissible electrical equipment for use at the working face is being more generally adopted, and for the well-being of all concerned in Utah coal mines it is hoped that within a relatively short period of time all devices and equipment used therein will be of permissible types.

It is confidently believed that with a responsive industry, effective inspection, close supervision, rigid discipline, and safety education of the workers, explosions in the coal mines of Utah will be greatly reduced if not wholly eliminated, notwithstanding the fact that up to the present the fatality rate in Utah coal mines from explosions is the poorest of the States that produce coal.

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INFORMATION CIRCULAR

UNITED STATES BUREAU OF MINES

EXPLOSIONS IN COLORADO COAL MINES, 1883 TO 1932¹

By G. M. Kintz² and E. H. Denny³

INTRODUCTION

In safety work a study of past accidents is of value in determining means of reducing future accidents of the same character. This is particularly true with reference to coal-mine explosions. Data from publications of the United States Bureau of Mines and from published reports of the State inspector of coal mines of Colorado are presented in this circular in the belief that they will be of value to operators by indicating methods of preventing future coal-mine explosions. The tables included are complete only with regard to major disasters. The Bureau of Mines considers as a major disaster one in which five or more men are killed. However, this circular also includes a considerable number of minor disasters. Various explosions in the coal mines of Colorado from 1883 to 1932, inclusive, are listed with information available as to their probable cause, number of men killed, and measures which, if they had been in use at that time, might have prevented them.

John McNeil, first Colorado State coal-mine inspector, on page 43 of his first report for the period ended July 31, 1884, made the following recommendations for the prevention of explosions in the coal industry:

True causes could be traced either to insufficient ventilation, carelessness and lack of rigid discipline in carrying on energetic ventilation and other precautions needed to be enforced in a mine giving off explosive gases. *** When any quantity of fine coal dust is observed in an entry, air course, or room it should be made damp and be removed from the mine, and thus keep every place as free from dust as possible. *** A system of bratticing close up to the face of each and every place in the mine from crosscut last made (whether gas should just then show its presence or not) which should not be more than 50 or 75 feet according to the nature of the mine.

He recommended noninflammable stoppings, "air bridges," and many other gas-ignition prevention measures that are used today, with the exception of permissible equipment, electric lamps, and rock-dusting. He also suggested the use of lime cartridges in place of black blasting powder.

Explosions still occur despite these early recommendations and the adoption by many operators of better ventilation and blasting methods, rock-dusting, and more modern coal-handling appliances. The importance of adequate ventilation, rock-dusting, the use of permissible explosives in a permissible manner, the use of permissible mining equipment in gassy mines, and thorough testing for the presence of inflammable gas is generally understood by the industry. Few mines, however, have completely carried out all means of explosion prevention, and as long as one or more of these factors is neglected explosions will still occur.

Table 1 gives available data regarding the cause, number killed and injured, and possible means of prevention at the time of 119 explosions that occurred in Colorado from 1883 to 1932, inclusive.

¹ The Bureau of Mines will welcome the reprinting of this paper, provided the following footnote acknowledgment is used: "Reprinted from U.S. Bureau of Mines Information Circular 6753."

² District engineer, U.S. Bureau of Mines, Denver, Colo.

³ Associate mining engineer, U.S. Bureau of Mines, Denver, Colo.

TABLE 1.- Explosions in Colorado Coal Mines, 1883 to 1932

Case No.	Date	County	Type of opening	Tons per year	No. employed	No. in mine	Extent of explosion	Cause of presence of explosive atmosphere	Possible source of ignition	Possible means of prevention at time	Gas or dust explosion	No. killed	No. injured	No. escaping uninjured
1	11/29/1883	Gunnison	Slope	47,524	75	-	Local	Feeder at face. Door left open short-circuited air	Open light	Some positive means of insuring doors are closed. Use of air bridge	Gas	1	3	71
2	1/2/1884	Huerfano	do.	17,976	50	-	do.	Poor ventilation of working place. Brushing out gas	Brushed gas out of room with coat onto open light on entry	Brattice cloth. Inspection. Forbid brushing out gas	do.	0	1	2/49
3	1/24/1884	Gunnison	do.	47,524	75	-	Entire mine	Broken-down brattice at face of room	Open light	Not moving gas with men in mine. Sprinkling	Gas and dust	59	1	10
4	1/12/1885	Fremont	do.	1/129,099	-	-	Local	Gas in abandoned area	Men with open lights disregarded "danger board" and entered gassy area	Inspector recommended discipline and that fire boss eradicate gas by proper ventilation	Gas	1	4	-
5	1/14/1888	do.	Shaft	-	335	-	do.	Fired shot, liberated gas	Open light	Better ventilation	do.	1	4	-
6	1/19/1888	do.	do.	-	335	-	do.	-	do.	Better ventilation	do.	0	1	-
7	2/9/1888	Park	Slope	23,421	80	-	do.	-	do.	Better ventilation	do.	0	1	-
8	4/25/1888	Las Animas	Drift	302,000	236	-	do.	-	do.	Better ventilation	do.	0	1	-
9	7/10/1888	Fremont	Slope	1/154,520	158	-	do.	-	do.	Better ventilation	do.	0	1	-
10	11/5/1888	Las Animas	Drift	181,270	204	2	Entire mine	Dead-end entry filled with gas. No line brattice	Open light. Explosion set off 6 kegs of black blasting powder and a box of 30% dynamite	Inspector recommended sprinkling of all workings. Only small amount of explosives to be kept in mine	Gas and dust	2	0	0
11	12/3/1888	Garfield	do.	Being developed	Unknown	0	do.	Dust on floor, ribs, and roof raised by shots	Blown-out shot	Inspector recommends sprinkling with water under pressure	Dust	4/3	4/8	0

1/ Includes production of 2 mines.

2/ Estimated.

TABLE 1.- Explosions in Colorado Coal Mines, 1893 to 1932 - Cont'd.

Case No.	Date	County	Type of opening	Tons per year	No. employed	No. in mine	Extent of explosion	Cause of presence of explosive atmosphere	Possible source of ignition	Possible means of prevention at time	Gas or dust explosion	No. killed	No. injured	No. escaping uninjured
12	12/10/1888	Fremont	Slope	3/154,520	158	11	Entire mine	Gas leaked from sealed-off area while fighting fire	Open lights	Use of flame safety lamps	Gas	2	9	0
13	9/3/1889	Garfield	Drift	49,147	150	-	Local	-	do.	Better ventilation	do.	0	1	Unknown
14	1/29/1890	Park	Slope	35,093	123	-	do.	Lack of line brattice caused gas to accumulate in working place	Open lights. Miner brushed out gas with shirt onto other miners' open lights	Inspector recommended use of line brattices and no open lights in section	do.	1	3	do.
15	3/29/1890	do.	do.	35,093	123	-	do.	-	Open lights	Better ventilation	do.	0	2	do.
16	10/24/1891	Garfield	Drift	11,211	60	-	do.	-	do.	do.	Gas & Dust	2	0	do.
17	12/18/1891	Park	Slope	52,626	150	-	do.	-	do.	do.	Gas	0	1	do.
18	9/28/1892	Huerfano	do.	69,959	120	-	do.	-	do.	do.	do.	2	-	-
19	10/6/1892	Garfield	Drift	43,780	70	-	do.	Dust raised by shots	Blown out shot	Sprinkling	Dust	1	4	-
20	1/10/1893	Park	Slope	-	-	-	-	-	Blown out shot of black blasting powder	Better ventilation, sprinkling	Gas and dust	24	-	-
21	1/22/1893	Fremont	Shaft	5,856	40	-	Local	-	Open lights	Better ventilation	Gas	0	2	-
22	6/22/1893	do.	do.	5,856	40	-	do.	-	do.	do.	do.	0	1	-
23	11/13/1893	do.	Slope	6/117,659	5/425	-	do.	-	do.	do.	do.	0	1	-
24	11/30/1893	do.	Shaft	5,850	40	-	-	-	do.	do.	do.	0	2	-
25	4/20/1894	Garfield	do.	83,693	200	-	-	Dust raised by shots	Blown-out shot ignited dust	Sprinkling	Dust	-	6	-
26	4/9/1895	do.	do.	114,923	148	-	Local	-	Open lights	-	Gas	-	1	-
27	5/10/1895	Las Animas	Slope	233,422	354	-	do.	Door propped open, short-circuiting air	do.	Better ventilation and discipline	do.	4	0	-
28	12/9/1895	Park	do.	41,881	111	-	do.	-	do.	-	do.	0	1	-
29	12/28/1895	Las Animas	do.	233,422	354	-	do.	-	do.	-	do.	0	1	-
30	2/18/1896	Garfield	do.	14,533	-	49	Entire mine	Unable to detect small quantities of gas with flame safety lamp	Uncertain; probably shot black blasting powder in chute	State inspector said there was nothing that equipment on the market at that time could do. He wished for better explosives and gas tester	Gas and dust	49	-	-

3/ Includes production of 2 mines. 5/ Includes 2 mines.

TABLE 1.- Explosions in Colorado Coal Mines, 1881 to 1932 - Cont'd.

Case No.	Date	County	Type of opening	Tons per year	No. employed	No. in mine	Extent of explosion	Cause of presence of explosive atmosphere	Possible source of ignition	Possible means of prevention at time	Gas or dust	No. killed	No. injured	No. escaping uninjured
31	2/29/1896	Park	Slope	33,887	106	-	Local	-	Open lights	-	Gas	0	1	-
32	3/23/1896	do.	do.	33,887	106	-	do.	-	do.	-	do.	0	1	-
33	4/16/1896	Las Animas	Drift	115,203	150	-	do.	-	do.	-	do.	0	1	-
34	3/24/1897	do.	Shaft	30,494	-	-	do.	-	do.	-	do.	0	1	-
35	6/9/1897	do.	Slope	156,936	176	-	do.	-	do.	-	do.	0	1	-
36	9/3/1897	Garfield	Drift	36,428	42	-	-	Dust raised by blasting	Uncertain; probably blown-out shot of black blasting powder	Shooting with only shot firers in the mine or using electric blasting	Dust	12	-	-
37	10/18/1897	Gunnison	Slope	233,903	311	-	Outside mine at tippie	Runaway trip down incline smashed through tippie causing dust cloud	Runaway trip upset stove on tippie	Automatic derailing	do.	0	2	-
38	1/8/1898	Las Animas	Shaft	10,770	50	-	Local	-	Open lights	-	Gas	0	1	-
39	6/30/1898	do.	Slope	171,535	176	-	do.	-	-	-	do.	0	1	-
40	4/21/1899	Huerfano	do.	113,853	150	-	-	Unknown	Unknown	Better ventilation	do.	0	1	-
41	4/21/1899	Garfield	Shaft	30,174	105	-	-	-	Open lights	do.	do.	0	2	-
42	4/27/1899	Las Animas	Drift	507,615	502	-	-	-	do.	do.	do.	0	1	-
43	11/12/1899	Garfield	Slope	34,603	40	-	-	-	do.	do.	do.	0	1	-
44	11/24/1899	Gunnison	Drift	15,503	60	-	-	-	do.	do.	do.	0	2	-
45	2/7/1900	Las Animas	do.	253,424	275	-	-	-	do.	do.	do.	0	2	-
46	2/18/1900	Huerfano	Slope	50,303	123	-	-	-	do.	do.	do.	0	1	-
47	3/31/1900	do.	do.	96,556	182	-	-	-	do.	do.	do.	0	1	-
48	4/14/1900	Garfield	do.	56,728	45	-	-	-	do.	do.	do.	0	1	-
49	4/23/1900	Gunnison	Shaft	73,430	90	-	-	-	do.	do.	do.	0	2	-
50	5/7/1900	Garfield	Slope	56,728	45	-	-	-	do.	do.	do.	0	2	-
51	5/21/1900	Las Animas	do.	255,179	300	-	-	-	do.	do.	do.	0	4	-
52	10/2/1900	Huerfano	do.	54,631	41	-	-	-	do.	do.	do.	0	1	-
53	10/20/1900	do.	do.	50,303	123	-	-	-	do.	do.	do.	0	1	-
54	1/8/1901	Garfield	Drift	Not listed	10	2	Entire mine	Improperly placed shots raised dust cloud. No gas	Blown-out shot at face of entry	Better placing of shots	Dust	2	0	0

TABLE 1.- Explosions in Colorado Coal Mines, 1883 to 1932 - Cont'd.

Case No.	Date	County	Type of opening	Tons per year	No. employed	No. in mine	Extent of explosion	Cause of presence of explosive atmosphere	Possible source of ignition	Possible means of prevention at time	Gas or dust explosion	No. killed	No. injured	No. escaping uninjured
55	1/27/1901	Pitkin	Slope	81,688	222	-	-	-	Open lights	Better ventilation	Gas	0	1	-
56	2/18/1901	Fremont	Shaft	86,576	70	-	-	-	do.	do.	do.	0	1	-
57	4/15/1901	Garfield	Slope	19,684	37	-	-	Improper ventilation	do.	Better ventilation; proper coursing of air	do.	0	1	-
								See case 61						
58	4/17/1901	do.	do.	19,684	37	-	-	do.	do.	do.	do.	0	4	-
59	5/9/1901	Boulder	Shaft	8,633	10	-	-	-	do.	Better ventilation	do.	0	2	-
60	5/21/1901	Las Animas	Slope	322,825	441	-	-	-	do.	do.	do.	0	1	-
61	5/31/1901	Garfield	do.	19,684	37	5	Entire mine	Too small an air course; improper method of coursing air	do.	Increase in size of air course. Proper inspection of mine before miners enter.	do.	6/2	17/7	0
62	9/16/1901	Gunnison	Drift	26,620	Not listed	-	-	-	do.	Better ventilation	do.	0	2	-
63	9/16/1901	Pitkin	Slope	214,643	137	3	Entire mine	Mine damp. Dust raised by blasting	Blown-out shot of black blasting powder	None recommended by State inspector	Dust	8/6	3	0
64	12/27/1901	Gunnison	Shaft	77,851	105	-	-	-	Open lights	Better ventilation	Gas	0	1	-
65	1/20/1902	Huerfano	Slope	137,401	153	-	Local	Cave at face of room nearly blocked crosscut	do.	Better ventilation	do.	3	2	More than 3
66	4/23/1902	Boulder	Shaft	5,270	10	-	do.	Door left open.	do.	Keeping crosscuts open				
								short-circuiting air		Double doors or air bridge (overcast)	do.	1	2	-
67	7/21/1902	Fremont	Slope	18,162	51	-	do.	-	do.	Better ventilation	do.	0	1	0
68	7/28/1902	Garfield	Rock tunnel	790	5	1	Entire mine	Dust raised by shots	Not certain. Probably blown-out shot	Sprinkling	Dust	1	0	0

6/ Miners killed outside of mine.

7/ Miners burned outside of mine.

8/ 3 miners outside of mine were killed.

TABLE 1.- Explosions in Colorado Coal Mines, 1883 to 1932 - Cont'd.

Case No.	Date	County	Type of opening	Tons per year	No. employed	No. in mine	Extent of explosion	Cause of presence of explosive atmosphere	Possible source of ignition	Possible means of prevention at time	Gas or dust explosion	No. killed	No. injured	No. escaping uninjured
69	8/2/1902	Las Animas	Shaft	Abandoned	5	5	Shaft	Old abandoned shaft known to be filled with gas	Not certain. Probably one of men working on pump rod in shaft struck a match to light pipe	More care in working around abandoned gassy mines	Gas	4	1	0
70	8/7/1902	do.	Drift	143,464	145	13	Entire mine	Mine nongassy but very dusty. Dust raised by explosives	Explosion of dynamite stored in mine	Sprinkling. Cleaning up of dust	Dust	13	0	0
71	9/2/1902	Huerfano	Slope	52,501	105	-	Local	-	Open lights	Better ventilation	Gas	0	2	0
72	7/11/1903	Garfield	Drift	48,632	34	1	Entire mine	Shots raised dust cloud. Mine very dusty although equipment and water for sprinkling were available	Blown-out shot. Dynamite	Discipline. Sprinkling	Dust	1	0	0
73	10/28/1904	Las Animas	do.	128,035	247	18	do.	Shots on solid. Miners not allowed to fire shots but did. Raised dust cloud	Blown-out shots	Better discipline. Shot firers were only ones allowed to fire shots. Miners disobeyed rules. Better location of holes	do.	19	0	0
74	2/19/1906	Huerfano	Slope	135,503	Not given	-	Local	Sudden outburst of gas	Open lights	Use only flame safety lamp in pillar workings	Gas	14	-	-
75	4/19/1906	Las Animas	Drift	241,152	-	-	do.	-	No details given	-	do.	1	-	-
76	4/22/1906	do.	do.	85,824	Not given	43	-	Room filled with gas after being examined and found clear	Open lights	Flame safety lamps. Better ventilation	Gas-dust	9/19	2	22

9/ Includes son visiting father.

TABLE 1.- Explosions in Colorado Coal Mines, 1883 to 1932 - Cont'd.

Case No.	Date	County	Type of opening	Tons per year	No. employed	No. in mine	Extent of explosion	Cause of presence of explosive atmosphere	Possible source of ignition	Possible means of prevention at time	Gas or dust	No. killed	No. injured	No. escaping uninjured
77	7/16/1906	Las Animas	Slope	105,714	-	-	Local	-	-	-	Gas	1	-	-
78	1/23/1907	do.	Drift	426,325	Est. 200+	24	Entire mine	Unknown. State inspector said possible for door to have short-circuited air	Unknown. May have been open safety lamp	Cause too uncertain to make definite recommendations	Gas-dust	24	0	0
79	4/24/1907	Huerfano	Slope	125,004	-	-	-	-	-	-	Gas	1	-	-
80	12/20/1907	do.	do.	111,410	-	-	Local	Shot on solid raised dust cloud	Blown-out shot. Shot on solid	Better location of drill holes	Dust	1	-	-
81	3/15/1908	Boulder	Shaft	105,873	-	-	do.	-	Probably open lights	Better ventilation	Gas	1	-	-
82	4/14/1908	La Plata	Tunnel	6,281	-	-	do.	-	Open lights	do.	do.	1	-	-
83	9/1/1908	Boulder	Shaft	37,545	-	-	-	-	do.	-	do.	1	-	-
84	11/9/1908	Mesa	-	-	-	-	-	-	-	-	do.	1	-	-
85	5/4/1909	Las Animas	Slope	304,305	-	-	Local	-	-	-	do.	1	-	-
86	7/6/1909	do.	Shaft	46,338	-	9	Entire mine	Stopping of fan over July 4 and 5	Open lights	Better ventilation by do. continued operation of fan	do.	9	-	-
87	11/14/1909	do.	Slope	273,167	-	1	Local	Dust raised by shots	Blown-out shot	Shots fired by electricity	Dust	1	-	-
88	12/14/1909	do.	do.	143,368	32	2	Entire mine	do.	do.	All shots fired by electricity from outside mine	do.	2	0	0
89	1/31/1910	do.	Drift	271,747	110	76	do.	Unknown	Uncertain	Sprinkling should be done more thoroughly	Gas-dust	75	0	1
90	10/8/1910	do.	do.	266,605	200	56	do.	Dust on timbers. Trip not sprinkled	Trolley trip split. Electric arc	Keep timbers free of dust and sprinkle sides and roof as well as road bed	Dust	56	0	-
91	11/8/1910	do.	Slope and 4 drifts	556,791	157	157	Local	Some persons claim explosion of CO. No gas found in mine	Haulage door caught on fire. Probably open lights	Use of overcast to eliminate doors on haulage roads	Gas-dust	10/79	-	78

10, Includes outsiders.

TABLE 1.- Explosions in Colorado Coal Mines, 1883 to 1932 - Cont'd.

Case No.	Date	County	Type of opening	Tons per year	No. employed	No. in mine	Extent of explosion	Cause of presence of explosive atmosphere	Possible source of ignition	Possible means of prevention at time	Gas or dust	No. killed	No. injured	No. escaping uninjured
92	2/9/1911	Las Animas	Slope	295,925	263	15	Local	Coal dust in gob. Excess of explosives in gob. Dust raised by shots	Blown-out shot	Proper storage of explosives; proper explosives	Dust explosion	10/17	-	2
93	5/4/1911	do.	do.	286,359	140	-	do.	Gas in abandoned workings	Open lights	Use of flame safety lamp	Gas	1	4	-
94	6/18/1912	do.	do.	231,468	198	43	Entire B seam	Deranged air current due to canvas "down" or door open	Defective safety lamp	Inspection and safe use of flame safety lamps	do.	12	1	30
95	8/29/1912	do.	do.	-	205	190	Local	Pocket of gas at roof Using carbide light and flame safety lamp at same time	Open lights	Better ventilation. Flame safety lamps	do.	2	0	4
96	10/1/1912	Garfield	Drift	32,514	71	-	-	Poor ventilation. No sprinkling. Dust raised by shots	Blown-out shot or open lights	Better ventilation and sprinkling	Gas-dust	2	0	-
97	11/8/1912	Boulder	Shaft	81,599	94	-	-	Gases distilled by fire	Fire ignited gases	-	Gas	1	4	-
98	12/16/1913	Garfield	Drift	76,096	48	37	Entire mine	Cloud of dust raised by fall of roof coal or rush of coal	Black blasting powder or open lights	Sprinkling	Dust	37	-	-
99	1/17/1914	Huerfano	Slope	51,555	91	-	Local	Blown-out shot and coal dust	Blown-out shot, on solid	Inspection and firing of shots by shot firers	do.	2	-	-
100	6/28/1914	Las Animas	Shaft	31,418	46	1	do.	Door left open	Relighting opened flame safety lamp	Proper ventilation. Proper handling of flame safety lamp	Gas	1	-	-
101	5/2/1915	Huerfano	do.	39,585	50	-	-	Air short-circuited through crosscut opened to remove rock for overcast	Open lights	Proper ventilating methods	do.	2	-	-

TABLE 1.- Explosions in Colorado Coal Mines, 1883 to 1932 - Cont'd.

Case No.	Date	County	Type of opening	Tons per year	No. employed	No. in mine	Extent of explosion	Cause of presence of explosive atmosphere	Possible source of ignition	Possible means of prevention at time	Gas or dust	No. killed	No. injured	No. escaping uninjured
102	4/27/1917	Las Animas	Slope	74,221	152	122	Entire mine	Probably due to short circuiting of air. Door left open or curtain "down"	Safety inspector trying to relight flame lamp with a match	Stricter supervision and magnetically locked flame safety lamps properly maintained and used	Gas-dust explosion	121	0	31
103	11/4/1918	Garfield	Shaft	36,638	43	-	-	Gas accumulated in rooms to pitch	Mine fire. Fire caused by spontaneous combustion	Proper ventilation	Gas	3	4	Unknown
104	3/31/1919	Las Animas	Slope	45,686	54	13	-	Fan stopped 2 hours on account of burned-out bearing	Short-circuit of electric feed wires	Cutting all power off mine when fan stops. Better installation of feed wires	do.	13	-	-
105	8/18/1919	Huerfano	do.	146,354	169	-	Entire mine	-	Defective flame safety lamp, blown-out shot, or matches	Properly maintained flame safety lamps. Use of permissible explosives. Obeying State laws	do.	18	-	-
106	2/12/1921	Routt	Drift	271,594	310	-	do.	Dust raised by shots	Blown-out shot of black blasting powder	Sprinkling at face. Use of permissible explosives	Dust	5	-	-
107	3/24/1922	Las Animas	Slope	262,094	175	17	do.	Locking door open short-circuited air	Nips of electric cutting machine	Overcast	Gas-dust	17	0	0
108	5/5/1923	do.	do.	32,831	54	10	do.	Fan stopped	Open lights	Permissible lamps; auxiliary power on fan	do.	10	0	0
109	10/7/1923	Mesa	Drift	18,056	34	7	do.	Poor ventilation. Fan operated intermittently. Not run at night	do.	Permissible lamps; continuous operation of fan	do.	6	-	1
110	8/5/1924	Huerfano	Slope	84,639	70	5	Local	Line brattices not extended to face. Poor installation	Igniting fuse with match	Electric blasting; proper installation of line brattice	Gas	1	0	4

TABLE 1.- Explosions in Colorado Coal Mines, 1893 to 1932 - Cont'd.

Case No.	Date	County	Type of opening	Tons per year	No. employed	No. in mine	Extent of explosion	Cause of presence of explosive atmosphere	Possible source of ignition	Possible means of prevention at time	Gas or dust	No. killed	No. injured	No. escaping uninjured
111	6/19/1925	Huerfano	Slope	47,236	30	-	Local	Cutting machine truck left under curtain	Electric arc	Proper ventilation; water on cutter bar	Gas	3	0	Unknown
112	1/23/1926	Garfield	Drift	Wagon mine	2	2	Entire mine	No fan. Mine not inspected regularly	Open lights	Positive thorough ventilation; permissible lamps	Gas-dust	3	0	0
113	1/29/1926	Las Animas	Slope	30,556	35+	35	do.	Insufficient ventilation of workings. Failure to drive crosscut at face before abandoning section	do.	Proper ventilation; permissible lamps	do.	4	6	25
114	5/27/1927	do.	do.	456,437	175	84	Local	Mine was partly rock-dusted but not sufficiently to prevent propagation of flame in certain places. Dust raised by wrecked trip	Runaway trip on slope broke power cable. Arc ignited dust raised by trip	Sprinkling trips; buried armored cable	Dust	7	-	77
115	2/5/1930	do.	do.	None	-	-	-	Abandoned	Unknown	Sealing of abandoned mines	Gas	0	0	0
116	7/13/1931	Gunnison	Drift	127,035	183	157	Outside on tippie	Dust cloud at tippie	Loose connection on crusher motor	Better wiring	Dust	0	2	-
117	7/23/1931	Las Animas	do.	55,470	89	1	Local	Insufficient and improper ventilation of rooms; no line brattice	Blown-out shot or spit of fuse	Electric shooting; proper ventilation	Gas-dust	1	0	-
118	10/23/1931	do.	do.	Abandoned	-	4	do.	No ventilation. Abandoned mine	Open lantern carried by experienced miner exploring old mine	Sealing abandoned mines	Gas	3	1	0
119	12/20/1932	Huerfano	Slope	58,832	122	67	do.	Leaking of gas from sealed area	Electric arc from open-type switch on hoist	Escaping gas carried directly into return air course	do.	1	1	65

As shown by table 2 there were only 7 years during the period of 50 years (1883 - 1932) without record of an explosion or gas ignition in the coal mines of Colorado, and only 13 years without record of a fatality due to an explosion.

TABLE 2.- Number of Explosions and Men Killed and Injured by Years in Colorado Coal Mines

Year	No. of explosions	No. killed	No. injured	Year	No. of explosions	No. killed	No. injured
1883	1	1	3	1909	4	13	-
1884	2	59	2	1910	3	210	-
1885	1	1	4	1911	2	18	4
1886	0	0	0	1912	4	17	5
1887	0	0	0	1913	1	37	-
1888	8	8	25	1914	2	3	-
1889	1	-	1	1915	1	2	-
1890	2	1	5	1916	0	0	0
1891	2	2	1	1917	1	121	-
1892	2	3	4	1918	1	3	4
1893	5	24	6	1919	2	31	-
1894	1	-	6	1920	-	-	-
1895	4	4	3	1921	1	5	-
1896	4	49	3	1922	1	17	-
1897	4	12	4	1923	2	16	-
1898	2	-	2	1924	1	1	-
1899	5	0	7	1925	1	3	-
1900	9	0	15	1926	2	7	6
1901	11	10	23	1927	1	7	-
1902	7	22	8	1928	-	-	-
1903	1	1	-	1929	-	-	-
1904	1	19	-	1930	1	0	0
1905	0	0	0	1931	3	4	3
1906	4	35	2	1932	1	1	1
1907	3	26	-	Total	119	797	147
1908	4	4	-				

NATURE OF EXPLOSIONS IN COLORADO COAL MINES

Records of 119 explosions and gas ignitions in Colorado coal mines have been compiled; however, 91 of these occurred before 1911. Their nature with reference to gas or coal dust is shown in the following table:

TABLE 3.- Nature of explosions with reference to gas or coal dust

Nature of explosion	No. of explosions	No. killed	No. injured
Gas.....	81	114	113
Dust.....	21	186	25
Gas and dust.....	17	497	9
Total.....	119	797	147

Of the 81 gas explosions listed, 46 occurred without killing anyone, and 1 occurred without injuring or killing anyone. The above table shows that nearly as many persons were injured as were killed in the gas explosions.

Of the 21 dust explosions listed men were injured in only 3 and no one was killed. The number of men injured was approximately 13.5 percent of the number of men killed.

Of the 17 gas and dust explosions listed not one occurred without killing someone; the number injured was approximately 1.8 percent of the number killed.

These figures indicate the necessity of thorough rock-dusting and of sprinkling places and trips where possible to prevent participation of dust in explosions in bituminous and lignitic coal mines. A gas and dust explosion is likely to be more disastrous than either a gas or a dust explosion alone; in fact, nearly all widespread coal-mine explosions have involved both gas and dust.

TABLE 4.- Sources of ignition

Cause	No. of explosions	No. killed	No. injured
Known to be open lights.....	26	187	54
Records not clear, probably due to open lights.....	53	12	57
Fires in mines.....	2	80	4
Flame safety lamps.....	5	176	1
Explosives.....	20	162	21
Electricity.....	7	97	3
Spontaneous combustion.....	1	3	4
Matches.....	2	5	1
Stove on tipple.....	1	0	2
Unknown.....	2	75	0
Total.....	119	797	147

Open lights.— Records show that open lights caused at least 26 explosions in which 187 men were killed and 54 injured. Incomplete older records indicate that open lights caused 53 additional explosions which killed 12 men and injured 57. The need of safe illumination for coal miners was recognized as long ago as 1815, when Davy and several others developed flame safety lamps. Over 20 years ago portable electric cap lamps were introduced into the coal mines, and now more than 250,000 permissible-type portable electric cap lamps are used in United States coal mines. Later-type lamps give as much or more light than carbide lamps, and it is probable that the yearly cost of electric lamps is less than that of carbide lamps.

The Bureau of Mines recommends that "in all coal mines the portable lamps for illumination be permissible, portable, electric mine lamps."⁴ The Bureau of Mines makes this recommendation from the standpoint of safety, not only on account of the gas-ignition hazard and the fire hazard of open-flame lamps but also because of the much greater dependability of up-to-date permissible electric cap lamps. In view of past experiences with the ignition of explosive gas by open-flame lamps and the demonstrated practicability of electric cap lamps, it is surprising that any coal-mine operators now allow open lights in their mines or that mine workers continue to use equipment known to be both unsafe and inefficient.

⁴ Bureau of Mines, Mine Safety Board Decision No. 1: Inf. Circ. 6732, Bureau of Mines, 1933, 43 pp.

Flame safety lamps.— Five explosions, killing 176 men, were probably due either directly or indirectly to improperly assembled or handled flame safety lamps. With a record of 176 of 797 deaths known to have occurred in Colorado coal mines the name "flame safety lamp" appears to be a misnomer, at least as applied to its experience in Colorado.

The Bureau of Mines has approved flame safety lamps for inspection purposes in gaseous mines, with the knowledge that they are safe to use in such mines if proper care is taken of them, but they are by no means "fool proof." One explosion was caused by a safety engineer taking his key-locked flame safety lamp apart underground and trying to relight it with a match; such an act is difficult to understand, but it explains in part why only magnetically-locked flame safety lamps should be used and why the lock should be kept in proper working condition and the magnet for opening the lamp should be available only outside the mine. In another explosion initiated by a flame safety lamp and causing the death of several people a fire boss was using a defective flame safety lamp which ignited some gas; this again shows that even in the hands of supposedly competent experts the lamp has its hazards.

Explosives.— The use of black blasting powder or dynamite or the improper use of permissible explosives caused 20 explosions resulting in the death of 162 men.

As early as 1898 there was on the market a so-called "Atlanta flameless powder" recommended by the Colorado State mine inspector for use in dusty and gassy mines.

In 1907 the first explosives now designated as "permissible" were used. Since that time explosives manufacturers have continually improved permissible explosives, and it is now possible to obtain permissible explosives that under intelligent usage will give virtually as effective blasting results as black powder. Explosives engineers have proved that the production of lump coal does not depend wholly upon the speed of detonation of an explosive and that strength and density are also important factors. The best method of determining the proper explosive to use in any coal is to consult the manufacturers, who have engineers well-qualified to recommend the grade of permissible explosives that should be used.

If permissible explosives are not used according to the methods prescribed by the Bureau of Mines and the manufacturers of explosives they may ignite an explosive mixture of gas or a coal-dust cloud. Even though the explosive is properly manufactured and approved by the Bureau of Mines it is not permissible if used in any of the following ways:

1. If stored under improper conditions until it undergoes a change in character.
2. If used frozen or partly frozen.
3. If used in excess of $1\frac{1}{2}$ pounds per shot.
4. If the diameter of the cartridge is less than that designated in the column "smallest permissible diameter."
5. If fired with an electric detonator of less efficiency than that prescribed.
6. If fired with fuse.
7. If fired without stemming.
8. If fired with combustible stemming.
9. If fired in the presence of a dangerous percentage of firedamp.
10. If the shot is a depending shot, is bored into the solid, or has a burden so heavy that the shot obviously is likely to blow out.

Some coal mines in Colorado continue to use black powder in blasting, and some mines in the State use permissible explosives but detonate them by means of fuse; electric detonators cost slightly more than fuse and detonators, but their safety is so much greater that the minor difference in cost should not be considered in comparison with the numerous safety advantages of electrical blasting.

Electricity.— Ninety-seven men lost their lives, and 3 were injured in 7 explosions in Colorado caused by electricity. With the increasing use of electricity in general in the United States there has been an increased number of gas ignitions and coal-dust explosions,

and even more than 7 explosions might have been expected in Colorado from this cause. The first explosion in this record due to electricity in Colorado occurred October 8, 1910, the last in December 1932. One explosion occurred at the tippie on the surface.

Marked improvement could be made in electrical installations in the coal mines of the State with regard to explosion and contact dangers. The haphazard manner in which electrical feed wires and electrical equipment are installed in many mines and tipples in the United States, as well as in Colorado, invites trouble from fires, is likely to lead to gas ignitions and explosions, and often offers utterly unnecessary man-contact hazards. In contrast, virtually all States have electrical building codes which, if applied to mines, would prohibit such installations as obtain in many if not most instances underground. These codes, moreover, apply only to buildings in which there ordinarily is no explosion hazard but only danger from fires and contacts.

Spontaneous combustion.— Numerous coal mines in Colorado are subject to spontaneous combustion. One fire from spontaneous combustion caused an explosion resulting in the death of 3 men and injury to 4 others. In the comparatively flat beds in northern Colorado, where spontaneous combustion of coal is frequent, the mining and ventilation methods developed have decreased the number of such fires and have usually made feasible their ready isolation and control. However, some steeply pitching coal beds in the State are subject to spontaneous combustion, and the control of fires in such beds is difficult and sometimes virtually impossible.

Fires in mines resulting in explosions.— While men were fighting the two mine fires listed as cases 91 and 97 explosions occurred which killed 80 men and injured 4. No gas had ever been found at the mine in case 91, nor was any found after the explosion. Some persons think the explosion was caused by the generation of carbon monoxide, others believe it was methane or other similar gas distilled from heated coal and coal dust. In this case exactly what set the door on fire is uncertain; open lights were used, and as the drivers usually ate their lunches by the door it is thought that a pile of wood chips, lunch papers, or the door itself caught fire from the flame of the open lights. It was while fighting this fire that the explosion occurred which killed 79 men, another instance indicating the hazards of open lights and of combustible doors, especially those on haulage roads in coal mines.

Matches.— Cases 69 and 110 are listed as due to matches. These two explosions caused the death of 5 men and injury of 1. In case 69 five men were working over an abandoned shaft which they knew was filled with explosive gas. It is believed that one of the men struck a match to light his pipe. In case 110 the shot firer lighted the fuses with a match.

Stove in the Tippie.— In case 37 a trip of cars broke away at the head of an incline and crashed into the tippie; the resulting dust cloud was ignited by an upset stove. Heating tippie houses with steam would remove such a source of ignition.

Unknown.— Cases 89 and 115 are listed as resulting from an unknown cause. Probably a blown-out shot of black blasting powder caused the explosion in case 89. The mine listed as case 115 had been abandoned for a number of years and stripped of all usable material but had never been sealed; the explosion probably was caused by some one outside the mine with an open flame. This mine was gassy and is the same one in which the disaster listed as case 108 occurred.

CAUSES OF GAS ACCUMULATIONS IN GAS AND GAS-AND-DUST EXPLOSIONS

Almost every gas accumulation is due to improper ventilation. Information is not available in every instance on the particular defects in ventilation at mines where the explosions listed occurred before 1910. Many lessons, however, can be learned by studying the defects given in table 5 that are known to have caused explosions.

TABLE 5.- Causes of gas accumulations in gas and gas-and-dust explosions

Cause	No. of explosions	No. killed	No. injured
1. Air short-circuited through doors or curtains.....	8	181	6
2. Fan stopped or was run intermittently....	4	38	0
3. Line brattices poorly installed or no line brattices.....	5	64	4
4. Machine truck left under curtain, short-circuiting the air.....	1	3	0
5. Abandoned mine.....	2	3	1
6. Abandoned or old workings.....	3	6	9
7. No fan, natural ventilation.....	1	3	0
8. Crosscut left open to remove rock for construction of overcast.....	1	2	0
9. Pocket of gas accumulating in high point in roof or workings driven to the rise..	2	5	4
10. Failure to drive crosscuts at proper intervals.....	1	4	6
11. Gas distilled by mine fire.....	2	80	4
12. Gas leaking from sealed area.....	2	3	10
13. Unable to detect less than 2 percent of methane on flame safety lamp.....	1	49	0
14. Gas feeder ignited by open lights.....	1	1	4
15. Improper ventilation.....	4	4	12
16. Cave caused accumulation of gas.....	1	3	2
17. Brushing out gas.....	1	0	1
18. Sudden outburst of gas.....	2	33	2
19. Uncertain.....	56	129	57
Total.....	98	611	122

Air short-circuited through door.-- It is almost impossible to operate a mine without some doors, but this is no reason why doors should not be used in pairs to form an airlock or should not be supplemented by curtains. Doors should be self-closing and should not be equipped with devices or attachments to hold them open. They should be fireproofed and so constructed that there is a minimum of air leakage.

Fans.-- In gassy mines, particularly those of considerable size, fans driven by electric motors should be so equipped with automatic starters that they will automatically start again if the power is interrupted; they should also be equipped with a device to shut off electric power in the mine in the event of fan stoppage. A mine fan should be run 24 hours every day; if a fan must be stopped for repairs all men should be removed from the mine, and the mine should be thoroughly examined by competent men before the electric current is turned on underground and before the other men are allowed to enter it.

Line brattice.-- Judging from the haphazard methods by which line brattices and curtains are installed in some mines it is apparent that many mine officials and men are loath to take such installations seriously. When properly installed and with enough air available line brattices are the best means of ventilating working faces when advancing between crosscuts or breakthroughs. However, there is leakage of air even with the best-installed curtains and brattices, and long line brattices are likely to be inefficient or even dangerous.

Equipment under curtains.— It will be noted that in one case an accumulation of gas with a resultant explosion is attributed to short-circuiting of the air by equipment, such as mining machines, cars, etc., left under curtains. Mine officials and workers should be alert to the avoidance of such practices, as they are unnecessary and obviously dangerous in the extreme.

Abandoned mines.— Two explosions, cases 115 and 118, killing 3 men and injuring 1, occurred in abandoned mines. The explosion in case 118 occurred while miners working in a gassy mine were exploring an abandoned mine in the same coal field in which their own mine was situated to determine whether they could mine out the old pillars. These men carried ordinary farm-type lanterns for light and should have known the extreme hazard of open lights in unventilated underground workings known or even likely to give off explosive gas.

Only a few explosions in abandoned mine are on record, but each year a number of persons are killed by ignition of explosive gas or suffocated by gases while exploring old mines that have not been sealed. When a mine is abandoned it should be effectively sealed, as required by law in many States.

Abandoned or worked-out areas.— Three explosions, resulting in the death of 6 men and the injury of 9 others, were caused by miners who entered an abandoned section of the mine while wearing open lights. In case 4 men stepped over a plainly marked "fire (or danger) board" to enter the gas filled region.

In the interest of safety the Bureau of Mines recommends that in coal mines all entries, rooms, panels, or sections that cannot be kept well ventilated throughout or cannot be inspected regularly and thoroughly, or that are not being used for coursing the air, travel, haulage, or the extraction of coal be sealed by strong fireproof stoppings.⁵

Natural ventilation.— To rely on natural ventilation for circulation of air through a coal mine is little short of criminal negligence, yet small mines often attempt to get along without fans. Efficient fans are now on the market at low cost, and if a mine operator cannot afford such a fan he should not be permitted to open or operate a coal mine.

Crosscut left open to remove rock.— Why any supervising official would permit men to work or electric power to be on the wires in by a point where the ventilating current had been short-circuited cannot be satisfactorily explained, yet this is done, at least in some instances with fatal consequences.

Gas accumulation in high points and in working to the rise.— On account of breaks and irregularities in the overlying strata high points often occur in the roofs of entries, air-courses, and rooms, as well as in pillar workings. Such places should be examined regularly for methane accumulations; if methane accumulates at such points they can be ventilated by brattice installed at the roof or by a pipe running from the high pocket directly into the return, thus passing the gas into the return airway, and in some instances it is discharged to the surface through boreholes suitably placed.

Where working places are driven to the rise in gassy mines officials should insist that the line brattices be kept close to the face and properly installed, also that crosscuts be driven at proper intervals and that there be enough air at the working faces to keep them clear of dangerous percentages of gas.

Crosscuts at regular intervals.— The Colorado mining law provides that crosscuts shall be driven as often as inspectors may order, but under no circumstances shall they be more than 60 feet in advance of the air current. Explosions have occurred because there was delay in driving such crosscuts, and this is particularly likely to occur in rise workings (crosscuts, entries, or rooms).

⁵ See footnote 3.

Gas from sealed areas.— Where possible, sealed regions should be located on the return air so that any gas leakage from such areas will not pass into live workings. Regular examinations should be made of the return air outby such seals to insure that an explosive mixture of inflammable gas is not present in the return, or in openings between the sealed region and the return.

Gas distilled from mine fires.— Mine fires are very likely to distill gases from the coal; these gases may be explosive, another reason why mines should take all feasible precautions against occurrence of mine fires and why mines should be equipped with fire-fighting apparatus underground. In both cases cited in this paper the fire could have been extinguished soon after discovery had such equipment been available.

Unable to detect less than 2 percent of methane with old safety lamps.— The State inspection department blamed some of the explosions from 1883 to 1898 on the fact that an instrument that would indicate the true percentage of gas was not available. The State mine inspector's biennial report for 1895-96 very definitely calls attention to the fact that few if any miners could detect on their safety lamps less than 2 percent and some not less than 4 percent of inflammable gas and that such percentages were too close to the explosive range for safety.

Permissible methane detectors that will record accurately less than 0.25 percent of methane are now on the market. Such detectors will tell mine officials much about the gassy conditions of their mines that they do not now know, and they will be of assistance in placing the ventilating currents where most needed; any mine in which explosive gas occurs should be provided with this up-to-date equipment and use it daily.

Feeders at working faces.— Where gas feeders occur extreme care should be taken to insure ventilation of the place so as to dilute the gas-air mixture below 0.50 percent. Open lights and black blasting powder certainly should not be used in mines wherein gas feeders occur or are likely to occur.

Improper ventilation.— Under this heading have been placed cases in which the air courses were known to be too small or had been allowed to fill up, or little or no effort had been made to send air into the working faces; conditions of these types are conducive to the accumulation of explosive gas. Some operators still do not realize the importance of clean air-courses of proper size or would rather gamble with an explosion than spend the money necessary to put aircourses in proper condition.

Brushing out gas.— The practice of brushing out gas has been condemned in Colorado ever since the first coal-mine inspector was appointed. In his first report he recommended for moving gas the use of brattice cloth, to be erected while all lights were out of the section.

Caves causing accumulations of gas.— Possible liberation of explosive gas when caving occurs is ordinarily anticipated by mine officials, and proper examinations are generally made of such caves. However, even small caves may liberate gas, and all caved areas near active workings should be examined as far as they are accessible during the regular fire-boss inspection.

Sudden outburst of gas.— This hazard (the sudden outburst of explosive gas) is present in some mines and can be handled with relative safety by proper ventilation and by the use of permissible equipment and permissible devices exclusively. The alertness and knowledge of the mine officials are great factors in avoiding all hazards in coal mines and are of especial value in the prevention of explosions and fires, and probably in no other phase of coal mining is the alertness and vigilance of officials more essential than in the avoidance of serious troubles in mines wherein outbursts of gas or dust, or both, may occur.

CAUSES OF DUST CLOUDS IN DUST EXPLOSIONS AND IN GAS-AND-DUST EXPLOSIONS

Rock-dusting prevents clouds of coal dust from being explosive, and thorough sprinkling of coal piles and loaded cars decreases the likelihood that a coal-dust cloud will be raised. Unless coal dust is actually wet (in general, this is difficult to accomplish) it may be raised into a cloud and ignited, causing an explosion, or may feed (or propagate) an explosion started elsewhere. The 38 explosions listed in table 6 illustrate the danger of coal dust when raised into a dense cloud and brought into contact with flame or an electric arc.

TABLE 6.- Causes of dust clouds in dust explosions and in gas-and-dust explosions

Cause	No. of	No.	No.
	explosions	killed	injured
Dust cloud raised by cave.....	1	37	0
Wreck of haulage trip (trip not sprinkled).....	2	63	0
Dust cloud raised by blasting.....	17	88	21
Dust cloud at tipple operations.....	2	0	4
Dust cloud raised by gas explosion.....	16	495	9
Total.....	38	683	34

Dust clouds caused by caves.— Caves, whether in active or in abandoned workings, besides liberating gas or causing gas accumulations have an additional hazard of raising a dense cloud of dust. Whether or not such clouds are explosive depends on their combustible content and density. Thorough rock-dusting of all accessible parts of mines, including abandoned open areas, will make such clouds noninflammable, and exclusion or restriction of open flames or flame-making equipment or devices will tend to prevent ignition of gas bodies or of dust clouds, even if they should occur.

Dust clouds caused by haulage.— Proper maintenance of haulage roads, thorough rock-dusting of haulage entries, and sprinkling of road beds and trips of cars would have prevented the explosions listed as cases 90 and 114.

Dust clouds from blown-out shots.— Improper location of drill holes, shooting off the solid, firing of dependent shots, use of too little or no stemming, and loading holes too heavily cause blown-out shots. These can be eliminated by proper supervision, assisted by safety education of miners and shot firers. Also, if the shots are loaded and fired when the working shift is out of the mine the likelihood of wholesale fatalities from explosions caused by blown-out shots or from other causes in connection with the use or abuse of explosives in coal mines will be eliminated.

Dust clouds caused at tipple operations.— Many coal men believe that the dust cloud raised by tipple operations is not dense enough to explode. The explosions listed as cases 37 and 116 show the fallacy of this opinion and the necessity of properly installing and maintaining electrical equipment and keeping tipples clean.

Dust clouds raised by gas ignition.— In Colorado as well as in nearly every State in which widespread explosions have occurred the usual causes have been an initial or local gas ignition, which usually affected only a relatively small area of underground workings, and an extension of the explosion through other parts of the mine or the entire mine by stirring the coal dust from the mine surfaces (floor, roof, ribs, and timbers) with accompanying ignition of the dust. Unless the immediate region in which the original gas ignition occurred was so wet that coal dust could not be stirred with the air or unless the dust was so thoroughly mixed with other incombustible (rock) dust as to be incapable of ignition and ex-

plosion the concussion from the original ignition of explosive gas was usually enough to cause the available dust from surrounding surfaces to be thrown into the air to feed the flame. This process usually was a continuing one if the mine workings were dry and rock-dusting had not been kept in effect efficiently.

The real solution of this type of explosion is the prevention of accumulations of explosive gas through proper ventilation; but this is not at all times feasible, and the problem is the more difficult since a very small quantity of an explosive mixture of gas and air (sometimes as little as 150 or 200 cubic feet) upon ignition can precipitate a widespread explosion if adjacent conditions as to dust, etc., are favorable. Hence, in addition to adequate ventilation and the elimination of open lights or other igniting agencies at or near faces or other places where explosive gas (and dust) may be expected it is extremely desirable that the surfaces of the immediate face region back about 40 or 50 feet be kept wet and that the surfaces of the remainder of the face region and the entire accessible open parts of the mine be kept well rock-dusted.

EXPLOSIBILITY OF COLORADO COAL DUST

Tests by the United States Bureau of Mines show that semianthracite, bituminous, and subbituminous coal dust is explosive under certain conditions. Anthracite dust is not thought to be explosive under ordinary conditions.

Experiments by the Bureau of Mines have proved that the ratio of volatile combustible to volatile combustible plus fixed carbon in the coal, expressed as V.M./V.M. + F.C., is an index of the explosibility of coal dust; the higher the volatile combustible matter the more explosive the dust, other factors being equal.

The following table shows this ratio, as calculated from available analyses, for coal from various counties in the State. (The ratio varies slightly for different mines.)

TABLE 7.- Ratio of volatile combustible to total combustible, by counties

County	Ratio, VM/VM + FC
Adams.....	0.402 to 0.475
Boulder.....	.396 to .482
Delta.....	.428
Elbert.....	.489
El Paso.....	.472 to .482
Fremont.....	.436
Garfield....	.409
Gunnison....	Bituminous .254 to .459
	Anthracite .034 to .078
Huerfano....	.431 to .446
Jackson.....	.433 to .515
Jefferson..	.469
La Plata....	.406 to .458
Las Animas	.380 to .445
Mesa.....	.419 to .431
Moffat.....	.463 to .486
Montezuma..	.264 to .510
Ouray.....	.479
Pitkin.....	.234 to .374
Park.....	.414
Rio Blanco	.386 to .501
Routt.....	.388 to .415
Weld.....	.381 to .542

Any coal dust that has a ratio of volatile to total combustible exceeding 0.15 is explosive. The presence of any quantity of inflammable gas in the mine air increases the explosibility of the dust, therefore an additional quantity of rock-dust must be added to neutralize the effect of the gas present.

For purposes of comparison, dust from the Pittsburgh seam in Pennsylvania, West Virginia, and Ohio has an average V.M./V.M.+F.C. of about 0.40. From table 7 it will be noted that dust from most of the bituminous coals of Colorado is at least as explosive as that from the coal of the Pittsburgh bed in Pennsylvania, and scores of tests of the Bureau of Mines at the Experimental mine, Bruceton, Pa., indicate that the dust of the Pittsburgh bed is among the most explosive dusts of the coal mines of the United States.

No coal-dust explosions of record have occurred in some counties in Colorado, particularly in the northern lignitic or subbituminous field. Probably the high moisture content of the coals in the northern field, supplemented considerably by good luck, has some bearing on this. The Bureau of Mines in its tests has initiated explosions in the absence of gas with coals having fully as high a moisture content as that in northern Colorado, and the past immunity from explosions is no guarantee that dust explosions cannot occur in mines producing these coals.

CONCLUSIONS

To have an explosion in a mine or on a tippie there must be an explosive mixture of methane and air, or a dense coal-dust cloud, or both, and enough heat must be present in the form of an open flame or an electric arc to ignite the mixture. Black blasting powder or dynamite when detonated may produce the necessary dust cloud and the flame may ignite the dust. When an explosion is initiated the advance wave may raise the dust cloud in front of the explosion flame and thus carry the explosion throughout the mine. Accumulations of explosive gas can be prevented by proper and thorough ventilation. Accumulations of explosive coal dust can be prevented by the application of enough rock-dust and by removing accumulations of coal-dust. Sprinkling of wall faces, coal piles, and trips of cars in a large measure prevents the air current from carrying dust throughout the mine. The possible sources of ignition can be decreased by the use of permissible equipment and permissible explosives, provided that the equipment and the explosives are maintained and used in a proper and safe manner.

Spontaneous combustion is a possible source of flame, and suitable mining methods can prevent spontaneous fires or can isolate and smother them if they occur; good ventilation will largely eliminate the danger of accumulation and ignition of gas from such a source.

The explosions listed in this paper did not just happen; they were caused by circumstances over which those engaged in coal mining had control. In recent years few lives have been lost in Colorado by coal-mine explosions compared with earlier years, undoubtedly due to better understanding by operators of coal-dust and gas hazards and to increased vigilance by the State mine-inspection department, as well as the operators and mine workers themselves. Nevertheless, the disastrous coal-mine explosion history of Colorado in the past and the fact that several minor explosions have occurred in recent years emphasize the need for continued vigilance on the part of all and the complete adoption and continued enforcement of explosion-prevention measures now well-known and readily available.

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UNITED STATES BUREAU OF MINES
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EXPLOSIONS IN KENTUCKY COAL MINES,
JANUARY 1, 1884 - JUNE 30, 1933

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BY

J. F. DAVIES AND H. B. HUMPHREY

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UNITED STATES BUREAU OF MINES

EXPLOSIONS IN KENTUCKY COAL MINES,
JANUARY 1, 1884 - JUNE 30, 1933¹

By J. F. Davies² and H. B. Humphrey³

PURPOSE OF THIS REPORT

Prevention of coal-mine explosions depends largely upon thorough knowledge of the causes that initiate explosions. A review of coal-mine explosions in one geographical area when supplemented by nationwide data of similar nature, should serve as a guide for the adoption of measures for explosion prevention. Explosions in bituminous-coal mines of the United States during the 10 years ended December 31, 1932, ranked third in the causes of fatal accidents and constituted 16.1 percent of all fatal accidents in bituminous mines.

Kentucky now ranks fourth in coal output in the coal-producing States of this country; a study of the coal-mine explosions in this State will be of definite interest locally among coal-mining people, as well as to those in the coal-mining industry who are concerned in preventing coal-mine explosions. This circular reviews gas and dust explosions in Kentucky coal mines to show their hazards, to point out the influence of certain factors in causing explosions, and to indicate the effectiveness of preventive measures.

The annual reports of the Kentucky Department of Mines and data published by the United States Bureau of Mines were used in compiling the tables of explosions and fatalities. These were supplemented by information furnished by individuals and operators connected with the industry in the State in past years. Through the cooperation of J. F. Daniel, chief of the Kentucky Department of Mines, several doubtful cases were clarified. So far as records are obtainable the history is comparatively complete, but it must be recognized that for various reasons some occurrences and fatalities

¹ The Bureau of Mines will welcome reprinting of this paper, provided the following footnote acknowledgment is used: "Reprinted from U. S. Bureau of Mines Information Circular 6754."

² Assistant mining engineer, U. S. Bureau of Mines, Norton, Va.

³ Assistant mining engineer, U. S. Bureau of Mines, Norton, Va.

were not reported or the records thereon were lost. Figures used in compiling some of the tables were furnished by W. W. Adams, chief statistician, Demographical Division, United States Bureau of Mines.

GROWTH IN PRODUCTION

Probably the first record regarding a coal deposit in the United States is that made in 1679 by Father Hennépin, a French Jesuit missionary, of a "cole mine" seen on the Illinois River near Ottawa, Ill., in what was then Kentucky. The first white settler to discover and use Kentucky coal was Dr. Thomas Walker. While on a survey journey on April 13, 1750, he found at a point within 2 miles of Cumberland Gap in what is now Bell County "blocks of very good coal on the surface". His diary records many other coal occurrences as he traversed the region. In 1790 the first iron furnace was erected near Owingsville, using charcoal for smelting but coal for refining the metal. Considerable amounts were used locally in the older towns, such as Lexington and Frankfort, after 1800.

In 1820 W. D. McLean opened what became known as the "McLean drift bank" on the Green River; this is regarded as the first commercial operation in the western coal field. In 1854 the Kentucky Geological Survey was established to gather and disseminate information on the extensive and important mineral deposits of the State. Coal production grew from 25,000 tons in 1840 to 100,000 tons in 1845 and 150,000 in 1850. Numerous wagon and barge mines were being worked along the Big Sandy, Licking, Kentucky, Cumberland, Green, and Tradewater Rivers. In the western field, Henderson County, a number of small surface mines were in operation, and to the south in Muhlenberg County the Mad River mine, opened in 1830, was actively producing. During and immediately after the Civil War production was greatly depressed, less than 170,000 tons being mined in 1870. By 1880 the production had reached a million tons annually, and it increased to over 2-1/2 million tons in 1890. Added transportation facilities helped to increase the figure to 35 million tons per year by 1920, and the greatest annual production was 69 million tons in 1927. In 1932, 35,610,000 tons production was reported.

HISTORY OF EXPLOSIONS

In the early years while the mines were preponderantly of the drift type and at no great distance from the surface the presence of explosive gas was noticed in only a few instances. The earliest ignition of which mention was found occurred in Garrard mine in Clay County in 1877; after the mine had been worked for some years gas appeared suddenly from a roof fissure, and in the resulting ignition one man and possibly others were seriously burned. A similar ignition occurred in the same mine in 1881, and there was another in the Mercer mine in Muhlenberg County about 1880. In 1885 gas was known to have appeared in only 6 mines, but the chief of the State Department of Mines issued repeated warnings that the hazard would increase as the workings were extended.

TABLE 1.--Gas and dust explosions in Kentucky coal mines, January 1, 1884, June 30, 1913

Case	Date	County	Mine, type	Approximate daily tonnage	Average No. in employed mine	Explosion		Cause	Source of ignition	Other factors	Possible means of prevention	Deaths	Injuries
						Type	Extent						
1	1887 Oct. 7	Hopkins.	Shaft, gaseous.	100	20	Gas, dust, strong.	Expansion.	Incomplete ventilation, gas under 5 percent, probably "windy" shot and 3 kegs of black blasting powder.	"Windy" shot.	Turned men.	Safer blasting, improve ventilations.	1	12
2	1888 Dec. 8	do.	do.	100	20	do.	Local.	Weak start.	Blasting.	Explosion.	do.	3	0
3	1894 -	do.	do.	200	50	Dust.	do.	Expansion.	Blow-out shot.	Black blasting powder.	Cut coal, safer blasting, and ventilation.	0	0
4	1895 Jan. 22	Union.	Slope, nongaseous.	25	5	do.	Strong.	do.	do.	Three kegs of black blasting powder.	do.	0	0
5	1897 Apr. 16	Hopkins.	Shaft, gaseous.	200	50	Gas, dust, violent.	Swept mine.	do.	do.	Shot fire.	Cut coal, safer blasting, and ventilation.	2	1
6	1899 Oct. 4	Union.	Slope, gaseous.	400	140	Gas.	Local.	No force.	Open light.	Entered against order.	Supervision.	0	0
7	1902 Feb. 15	Muhlenberg.	Shaft, gaseous.	1,000	200	Gas, dust.	do.	do.	Blasting.	Black blasting powder.	Ventilation, cut coal, safer blasting.	0	16
8	1903 Dec.	Whitley.	Drift, nongaseous.	-	-	Dust.	Strong.	Expansion.	Blow-out shot.	2 miners.	Cut coal, safer blasting, control dust.	2	-
9	1904 Apr. 20	do.	do.	50	11	do.	Violent.	To outside.	do.	3 on outside	Cut coal, safer blasting, water.	5	0
10	1904 Dec. 2	do.	do.	150	30	do.	Strong.	Expansion	do.	Overloaded.	Cut coal, ventilation, safer blasting.	1	6
11	1905 Mar. 3	do.	do.	100?	20?	do.	Violent.	To outside.	do.	Black blasting powder.	Cut coal, safer blasting.	0	2
12	1905 Dec. 29	do.	do.	200	50	do.	do.	do.	Blasting.	2 shot fire.	do.	2	1
13	1906 Mar. 10	do.	do.	100?	-	do.	Local.	Expansion.	do.	1 hurt outside.	do.	-	-
14	1906 July 12	Muhlenberg.	Slope, gaseous.	500	70	Gas.	Strong.	do.	Open light.	Dusty.	Cut coal, safer blasting, water.	2	-
15	1906 Oct. 2	do.	do.	500	70	do.	Ignition.	Small volume.	do.	Explosion in.	Ventilation and inspection.	0	-
16	1907 Jan. 21	do.	do.	300	50	do.	Local.	Expansion.	do.	Marked out.	do.	1	0
17	1907 Nov. 21	do.	do.	-	100	Dust.	do.	do.	Blasting.	Another Feb. 5.	do.	0	0
18	1907 Dec. 7	Bell.	Drift, nongaseous.	250	46	do.	do.	do.	do.	Dusty.	Safer blasting.	1	4
19	1908 Feb. 4	Union.	Shaft, gaseous.	250	50	do.	do.	do.	do.	Dusty mine, coal dust stemming.	Cut coal, ventilation, safer blasting, water.	0	8
20	1908 Feb. 10	Muhlenberg.	do.	75	20	do.	Strong.	do.	do.	do.	Cut coal, safer blasting, supervision.	3	4
21	1908 Oct. 9	Bell.	Drift, nongaseous.	30	13	do.	Local.	do.	do.	do.	Cut coal, safer blasting, water.	9	1
22	1908 -	Whitley.	do.	250	50	do.	do.	do.	do.	do.	Cut coal, safer blasting.	0	8
23	1909 Dec. 11	Webster.	Shaft, gaseous.	650	130	Gas.	Violent	Expansion, water.	Open fire in light.	Water came to surface.	Holes to release gas.	7	0
24	1910 Feb. 1	Muhlenberg.	do.	400	80	Gas, dust.	do.	Expansion.	Open light.	Spread by kegs of black blasting powder.	Ventilation, safer storage of powder, inspection, water.	3	6
25	1910 Feb. 8	Whitley.	Drift, gaseous.	400	150	Dust.	Strong.	do.	Blow-out shot.	do.	Cut coal, safer blasting, water.	6	0
26	1910 Nov. 25	Webster.	Shaft, gaseous.	100	20	do.	Violent.	Wrecked mine.	do.	Kegs of black blasting powder.	Cut coal, safer blasting, and storage.	10	0
27	1911 Jan. 6	Lee.	Drift, nongaseous.	75	20	do.	Local.	Expansion.	Blow-out shot.	Black blasting powder.	Cut coal, safer blasting.	0	17
28	1911 Jan. 21	Webster.	Slope, nongaseous.	650	125	do.	do.	do.	do.	Powder's smoke.	do.	0	0
29	1911 Dec. 8	Muhlenberg.	Shaft, gaseous.	75	15	do.	do.	do.	do.	Powder's smoke.	do.	1	0
30	1912 Jan. 17	do.	do.	750	150	do.	Strong.	Expansion.	do.	Aided by kegs of black blasting powder.	Cut coal, permissible explosives.	5	0
31	1912 Apr. 21	Hopkins.	do.	180	40	Gas.	Violent.	Wrecked mine.	Open light.	Powder.	Start fan before entering, closed lights, inspection.	5	0
32	1914 Jan. 1	Webster.	do.	500	100	Dust.	Strong.	Expansion.	Blow-out shot.	Keg black blasting powder.	Cut coal, permissible blasting, rock-dust.	1	1
33	1916 May -	do.	do.	500	100	do.	do.	do.	do.	Inadequate inspection.	Inspection, closed lights, rock-dust.	0	0
34	1917 Aug. 4	do.	do.	1,500	300	Gas, dust, violent.	do.	Expansion and inert dust, furnace ventilation.	Open light.	do.	Inspection, closed lights, rock-dust.	62	18
35	1917 Dec. 10	McCreary.	Drift, nongaseous.	300	75	Dust.	do.	Wrecked fan. Solid shooting, 2 shot fire.	Blow-out shot.	do.	Cut coal, permissible blasting, rock-dust.	2	0
36	1917 Dec. 30	Letcher.	do.	300	75	do.	Strong.	Expansion.	do.	do.	Permissible blasting, supervision, rock-dust.	0	4
37	1918 July 27	Union.	Slope, gaseous.	600	150	Gas.	Local.	do.	Open light.	Pumper went in before fire-boss.	Ventilation, inspection, closed lights.	1	0
38	1919 Feb. -	-	-	-	-	Dust.	-	-	No report.	-	-	2	-
39	1919 Dec. -	-	-	-	-	do.	-	-	do.	-	-	1	-
40	1920 Feb. 23	McCreary.	Drift, gaseous.	500	100	Gas.	Local	Small volume, ignition, um.	Open light.	Traces	Closed lights, supervision.	1	0
41	1920 Apr. 13	Harlan	do.	1,500	600	do.	do.	Inadequate ventilation, place marked out.	do.	Foreman and coal loader went in Safety Lamp, not lit.	Ventilation, closed lights.	1	0

1 No explosions, 1884-1886.

2 McCreary County established in 1912 from Whitley County.

TABLE 1.--Gas and dust explosions in Kentucky coal mines, January 1, 1884-June 30, 1933 (Continued)

Case	Date	County	Mine, type	Approximate daily tonnage	Explosion			Cause	Source of ignition	Other factors	Possible means of prevention	Injured	Fatalities
					Average No. in mine employed	Extent	Probably limited by						
42	1920 Dec. 20	Letcher	Drift, nongassy.	100	2 Dust.	Local.	Expansion.	Solid shooting.	"Windy" shot.	-	Safer blasting, cut coal, rock-dust.	2	0
43	1921 Sept. 12	Muhlenberg	Shaft, gassy	500	10 Gas.	do.	do.	Fireboss moving gas by brushing.	Open light.	Had mixed lights.	Ventilation, closed lights.	1	0
44	1921 Oct. 17	Pike	Drift, nongassy.	100	10 Dust.	do.	do.	Several solid shots, black blasting powder.	Blown-out shot.	Coal-dust tamping.	Cut coal, permissible blasting.	1	0
45	1922 Jan. 30	Bell	do.	150	8 do.	To outside.	Weakness.	Dust raised by 5 other solid shots.	do.	Furnace was out.	Cut coal, permissible blasting, ventilation, supervision.	6	2
46	1922 Feb. 7	Pike	do.	275	11 do.	do.	do.	Dust raised by 5 other "windy" shots.	do.	Overcharged, on solid.	do.	9	2
47	1922 Mar. 23	do.	Drift, gassy.	210	- Gas.	Local	No force.	At room face.	Open light(?)	Burned.	Inspection and closed lights.	1	0
48	1922 Apr. 3	do.	Drift, nongassy.	100	3 Dust.	do.	Weakness.	Dust raised by 8 solid shots.	Blown-out shot.	Had been ordered out.	Cut coal, permissible blasting, supervision.	2	0
49	1923 Feb. 23	Muhlenberg	Shaft, gassy.	1,600	75 Gas.	do.	Expansion.	Opening sealed area.	Open light.	Lack proper ventilation and inspection.	Ventilation, inspection, closed lights.	1	7
50	1923 Mar. 29	Union	Slope, gassy.	500	1 do.	do.	No force.	Went over inspector's danger board.	do.	Burned.	Closed lights and supervision.	1	0
51	1923 June 1	do.	Drift, gassy.	1,200	1 Dust.	do.	Weak start.	Solid shots, black blasting powder.	do.	-	Ventilation and closed lights.	1	-
52	1923 Dec. 1	McCreary	Drift, gassy.	1,200	1 Dust.	Local.	do.	3 solid shots, 1 "windy", 1 blown-out.	Blown-out shot.	Coal-dust stemming.	Cut coal, permissible blasting.	1	0
53	1923 Dec. 7	Perry	Drift, nongassy.	150	16 Gas, dust.	Strong.	Expansion.	8 solid shots in room and croquet.	do.	-	Cut coal, permissible blasting, rock-dust.	9	4
54	1924 Oct. 22	Hopkins	Shaft, nongassy.	1,000	18 Dust.	Violent.	Expansion only.	Machine shot on solid this day, overcharged.	"Windy" shot.	2 kegs black blasting powder.	do.	3	0
55	1925 Jan. 15	Webster	Slope, nongassy.	500	7 do.	Strong.	Damp dirt on slope entries.	Repair crew.	Blown-out shot.	Black blasting powder, coal dust.	Cut coal, permissible blasting, rock-dust, water.	6	0
56	1925 May 3	Hopkins	Shaft, gassy.	500	7 Gas.	Local.	No force.	do.	Open light.	Burned.	Ventilation, inspection, closed lights.	2	3
57	1925 June 8	Union	do.	1,000	180 Gas, dust.	do.	Rock-dust.	Fireboss "passed up" rooms.	do.	Confined to section not rock-dusted.	do.	17	0
58	1925 Sept. 1	Muhlenberg	do.	300	50 Gas.	do.	No force.	Room fell in, gas accumulated.	do.	Olimbed on fall.	Closed lights, inspection.	2	1
59	1925 Nov. 18	Hopkins	Shaft, nongassy.	200	7 Dust.	Strong.	Expansion.	Black powder in keg ignited by open light.	do.	2 in safety chamber.	Closed lights, permissible blasting, out coal.	5	0
60	1926 Feb. 16	Muhlenberg	Shaft, gassy.	500	100 Gas, dust.	Local.	Inert dust, weak start.	Dependent shot on solid, black blasting powder.	Blown-out shot.	3 killed by explosion, 5 rescuers by CO.	Cut coal, permissible blasting.	8	0
61	1926 Apr. 21	Webster	do.	1,500	2 Gas.	Local.	No force.	Fireboss ran into gas.	Open light.	-	Closed lights.	1	0
62	1927 Jan. 31	Hopkins	Slope, nongassy.	100	10 do.	do.	Small force and wetness.	First appearance of gas, fan idle over night.	Electric arc.	When power put on blower fan.	Ventilation and inspection before entrance of men.	4	0
63	1927 Mar. 14	Bell	Drift, gassy.	1,100	1 do.	do.	Small force.	Unventilated face, fireboss ran mining machine.	Open light.	Section foreman.	Ventilation, inspection, closed lights.	1	0
64	1927 Aug. 3	Webster	Shaft, gassy.	900	45 Gas, dust.	do.	Rock-dust.	Gas liberated by shots, passed over mining machine.	Electric arc.	Open-type machine.	Ventilation, inspection, rock-dust, permissible equipment.	15	0
65	1927 Oct. 29	McCreary	Drift, gassy.	900	200 Gas.	do.	No force.	"Fanning" out gas, passed over lights.	Open light.	Burned.	Ventilation, closed lights.	1	0
66	1928 May 22	Harlan	Drift, nongassy.	3,300	66 Dust.	Strong.	Expansion.	Adobe shot on large rock.	Blasting.	40 percent dynamite, 1 killed in recovery work.	Permissible blasting, rock-dust, water on machine.	8	0
67	1928 Nov. 27	Martin	Drift, gassy.	1,100	3 Gas.	Local.	do.	Mine idle since July.	Open light.	3 officials.	Closed lights, supervision.	3	0
68	1928 Dec. 18	Muhlenberg	Shaft, gassy.	1,000	11 Gas, dust.	do.	do.	Caving rooms liberated gas.	do.	Opened flame safety lamp.	Ventilation, inspection, closed lights, rock-dust.	6	0
69	1929 Jan. 21	Hopkins	Slope, nongassy.	200	2 Dust.	Wrecked mine.	Small mine.	Incompetent supervision.	Blown-out shot.	Black blasting powder.	Supervision, cut coal, permissible blasting.	2	0
70	1930 Mar. 29	Bell	Drift, gassy.	1,100	200 Gas, dust.	Strong.	Wet section, expansion.	Partly sealed area.	Open light.	Entered unventilated area with open light.	Ventilation or sealing, inspection, closed lights, rock-dust.	16	0
71	1931 Oct. 29	Harlan	do.	4,500	500 Gas.	Local	Wet area, weak ignition.	Machine cut feeder, lack of ventilation.	Smoking.	Killed by burns.	Ventilation, inspection, prohibition of smoking.	3	1
72	1932 Dec. 9	do.	Drift, nongassy.	250	37 Dust.	Strong.	Expansion.	2 blown-out solid shots, black blasting powder.	Blown-out shot.	Cobbed fines.	Permissible blasting, cut coal, rock-dust, supervision.	23	0
73	1933 May 15	McCreary	Drift, gassy.	600	125 Gas.	Local.	Inert dust.	Inadequate inspection and ventilation.	Open light.	Engineer party.	Ventilation, inspection, closed lights.	3	0
Total				41,195	7,988	3,127						344	123

Total

41,195 7,988 3,127

Note: In the years which are omitted no explosions or fatal ignitions are known to have occurred.

Table 1 lists 73 explosions in Kentucky coal mines of which record was available from 1884 to June 30, 1933, inclusive. Except for a few cases where full information was lacking the essential facts about the explosions are given; these explosions include all in which fatalities occurred or in which there was appreciable violence. Table 2 lists 42 other ignitions in which no fatalities resulted and no violence was noted but which caused burns or other injuries to 66 men; it is certain that many other such instances passed without being reported, hence this list must be regarded merely as an exhibit of potential disasters which were escaped by fortunate chance, as protective measures were so few, that it is questionable if they aided in limiting the explosions.

TABLE 2. - Nonfatal ignitions of gas or dust, 1877 - 1932.

Year	County	Circumstances	Source of ignition	Men injured
1877	Clay.	Feeder from roof fissure.	Open light.	2 miners.
1880	Muhlenberg.	Unventilated.	do.	Owner.
1881	Clay.	Feeder from roof fissure.	do.	1 miner.
1887	Hopkins	Insufficient ventilation.	do.	2 miners.
1893	Union	Unventilated section.	do.	Driller.
1894	Lee.	Powder and dust.	Overcharged shot.	3 miners.
1896	Whitley.	Powder smoke, dust, and gas.	Black powder shot on solid.	1 miner.
1898	Bell.	Feeders and insufficient ventilation.	Open light.	Superintendent.
1899	Muhlenberg.	Insufficient ventilation and inspection.	do.	Bank boss.
1901	Hopkins.	Dust and black powder.	Blown-out shot.	3 miners.
1901	do.	Poor ventilation.	Open light.	2 miners.
1903	Union.	Insufficient ventilation and inspection.	do.	do.
1904	Hopkins.	Old unventilated room.	do.	1 miner.
1904	Muhlenberg.	Gas, dust, powder smoke.	Blown-out shot.	2 miners.
1904	Union.	Poor ventilation.	Open light.	3 miners.
1904	Muhlenberg	Unventilated fire area.	do.	do.
1905	Hopkins.	Poor ventilation.	do.	1 miner.
1906	do.	Abandoned unventilated room.	do.	do.
1906	Knox.	Powder smoke, gas, and dust.	Blasting.	3 miners.
1906	Muhlenberg.	do.	do.	1 miner.
1907	Carter.	Poor ventilation and black blasting powder.	Blown-cut shot.	3 miners.
1907	Knox.	Powder smoke, gas, and dust.	Blasting.	4 miners.
1907	Whitley.	Dust and powder.	"Windy" shot.	2 miners.
1909	Hopkins.	Unventilated idle heading.	Open light.	1 driver.
1909	Muhlenberg.	Shot 11 holes near one another, smoke and dust.	Blasting.	1 shot firer.
1910	do.	Feeder and poor ventilation.	Open light.	1 miner.
1911	Whitley.	Insufficient ventilation, no inspection.	do.	do.
1913	McCreary.	Overfall, brushed out.	do.	Superintendent.

TABLE 2. - Nonfatal ignitions of gas or dust, 1877 - 1932 (Continued)

Year	County	Circumstances	Source of ignition	Men injured.
1913	McCreary.	Gas in room, brushed out.	Open light.	1 miner.
1919	do.	Entry head, insufficient ventilation.	do.	2 miners.
1920	do.	Insufficient ventilation, entry head.	do.	1 miner.
1920	do.	Insufficient ventilation and inspection of entry.	do.	do.
1921	do.	do.	do.	do.
1922	Bell.	Insufficient ventilation, no inspection.	do.	2 miners.
1924	McCreary.	Insufficient ventilation.	do.	1 tool nipper.
1924	Floyd	do.	do.	1 miner.
1926	McCreary.	Insufficient ventilation and inspection.	do.	2 shot firers.
1927	do.	Idle room, gas known, not marked.	do.	1 miner.
1928	do.	Insufficient ventilation and inspection.	do.	do.
1928	Martin.	Insufficient ventilation, nongassy.	do.	do.
1928	McCreary.	Insufficient ventilation and inspection.	do.	1 machine man.
1929	do.	Insufficient ventilation and inspection of entry.	do.	do.

TABLE 3. - Explosion fatality rates and percentage of total for different periods

Period, years	Fatality per million		Percent of total no. killed	
	Tons	Man-hours (inside)	Kentucky	United States
1884-1890 (7 years)	0.43	0.10	11.8	9.0 (est.)
1891-1900 (10 years)	.19	.05	6.5	15.9
1901-1910 (10 years)	.90	.31	23.5	30.0
1911-1920 (10 years)	.35	.17	10.9	12.4
1921-1925 (5 years)	.32	.22	9.7	14.8
1926-1930 (5 years)	.21	.14	7.0	12.5
1931	.08	.06	2.7	6.0
1932	.65	.57	22.8	14.3
Total, 1884-1932	- -	- -	11.0	14.0

FATALITIES⁴

Available records, as listed in table 1, indicate that in the 49-1/2 years from January 1, 1884, to June 30, 1933, 344 fatalities occurred in Kentucky coal mines as a result of 73 explosions of gas and dust, an average of 4.7 fatalities for each explosion. In 12 explosions none was killed. The greatest number of deaths in a single explosion was 62, and the next highest number was 34. Only 6 explosions claimed more than 10 victims. The records show 123 men injured more or less seriously in these 73 explosions. It is notable that few escaped such disasters with injury and that in the last 15 explosions only 1 injured man survived.

Table 3 presents the gas and dust fatality rates per million tons mined and per million man-hours of underground exposure. These rates parallel closely those of some older coal-producing States. The high tonnage rating up to 1910 is usual, reflecting the low tonnage per man-day of that period. The drop from this point may be attributed to two influences--greater tonnage per man and safer practices introduced to reduce explosion hazards. The rating on the man-hour basis is the more significant from the viewpoint of safety to the men. The rates of 0.17 fatality per million man-hours from 1911 to 1920 and 0.22 from 1921 to 1925 are due to a long series of large and small explosions coincident with an increase in production from 14 to 60 million tons, with an increase in man-hours from about 35 to 80 millions.

⁴ The number of explosions and men killed differ from yearly totals given by the Bureau of Mines and the State due to the inclusion of fatalities segregated by them under other headings and some in mines not under State control.

The average rates for the 5-year period from 1926 to 1930 were 0.21 per million tons and 0.14 per million man-hours. The corresponding rate per million man-hours for the United States was 0.31. In 1931 only 3 fatalities occurred from this source in Kentucky, giving fatality rates of 0.08 and 0.06 per million tons and man-hours, respectively. These rates jumped to 0.65 and 0.57, respectively, in 1932, when 25 men were killed in 1 explosion. In 1932 fatality rates for the country as a whole were 0.47 per million tons and 0.25 per million man-hours.

Gas-and-dust explosion fatalities in Kentucky averaged 10 percent of the total number killed in coal mines of the State over the entire period, 1884 to June 30, 1933, inclusive. In comparison, those killed in explosions in coal mines of the entire United States for this period were about 14 percent of the total; it is possible that this ratio indicates approximately the relative hazard from this source for Kentucky coal mines compared with coal mines of the United States as a whole.

CAUSES OF EXPLOSIONS

Three general causes are given for explosions in Kentucky. The majority, 42, were caused by so-called "windy" or blown-out shots, 27 by open lights or flames, and 3 by electric arcs. In 2 others no details are available, but in all probability blasting was the cause.

TABLE 4. - Causes of Explosions in Kentucky coal mines,
January 1, 1884, to June 30, 1933.

Causes	Explosions	Percent
Blasting	42	57
Open lights or flames	27	37
Electric arcs	3	3
Not given	2	3
Total	73	100

Blasting. - In the 49-1.2 year period 57 percent of the explosions listed in table 1 were caused by blasting. From 1884 to 1917, inclusive, there were 28, or 73 percent of a total of 36 explosions from blown-out or "windy" shots; from 1918 to 1927, inclusive, there were 11 explosions from this cause, or 38 percent of the 29 explosions which occurred during this period. Three, or 40 percent of the 8 explosions occurring from January 1, 1928, to June 30, 1933, were from blasting.

TABLE 5. - Summary of conditions attending explosions caused by blasting in Kentucky coal mines

Causes	Explosions
Solid shooting, blown-out or "windy" shot, coal-dust stemming, igniting accumulated explosives, smoke and dust	7
Solid shooting, blown-out or "windy" shot, coal-dust stemming, igniting dust	21
Solid shooting, blown-out or "windy" shot, coal-dust stemming, igniting gas and dust	6
Solid shooting, blown-out or "windy" shot, coal dust stemming, igniting kegs of black blasting powder and dust	6
Dynamite on large rock, dust explosion	1
Keg of black blasting powder ignited by open light	1
Total	42

Except for the case where dynamite was placed on a large rock to break it, all explosions by blasting were caused by shooting coal off the solid or using black powder in keg or pellet form. In 21 cases dust was raised into a cloud and ignited by black blasting-powder shots. In 7 other cases accumulated explosives smoke and dust were blamed; in 6 more cases concentrations of gas and dust were ignited; and in the other 6 cases kegs of black blasting powder stored in the workings were ignited and helped to propagate dust explosions. Coal-dust stemming helped to ignite the dust cloud in numerous instances despite admonitions of the assistant inspectors, repeated year after year, to use clay or other incombustible material. The greatest hazard lies in the practice of shooting from the solid with little or no supervision. Had it not been for the introduction in certain mines of regulations requiring shooting to be done by shot-firers after the men on the regular shift were out of the mine the loss of life undoubtedly would have been much greater. Lack of adequate ventilation, which allowed dust, gas, and smoke to hang in clouds, was a contributing factor in numerous instances. The dominant cause was the use of black blasting powder, first in keg form and more lately in the pellet or stick form. Two disastrous dust explosions occurred in mines in which the coal was cut by machine and in which some shooting from the solid was also allowed; and in 40 of the 42 explosions caused by blasting, solid shooting was in effect.

Open lights.- Twenty-seven (37 percent of the 73 explosions listed in table 1) were caused by open lights, including all types of open flames. Except in 2 cases, open-flame miners' lamps were the source of the ignitions;

ignition in one mine was charged to the boiler fires or other open flame on the surface and in the other to smoking at the face.

From 1884 to 1917, inclusive, 8 (22 percent) of the 36 explosions listed were caused by open lights or flames. In the next 10 years, from 1918 to 1927, inclusive, 14 (50 percent) of the explosions fell in this class. From 1928 to June 30, 1933, 30 percent of the explosions were initiated by open flames.

TABLE 6. - Summary of conditions attending explosions caused by open lights in Kentucky

Causes	Explosions
Carrying open light into unventilated, uninspected, working place	14
Carrying open light into marked-out place	5
Brushing out gas in presence of open light	2
Use of open light while inspecting for gas	4
Smoking at unventilated face	1
Outburst of gas reached open flame at surface	1
Total	27

The use of open lights in mines which in the majority of cases were known to liberate gas was also concomitant with other notable lapses from safe practices in all but the last instance. These unsafe practices frequently violated warnings and instructions issued by the State inspectors. Neglect or refusal to conform with recognized standards is evidence of lack of supervision or of inefficient supervision.

Electric arcs.- The 2 explosions attributed to electric arcs occurred in the same year, 1927. In one explosion power had been cut off the mine, and the fan had been idle overnight. Arcing from a blower-fan motor inside the mine ignited accumulated gas when the main switch was put in the next morning. The second explosion occurred when gas liberated by blasting passed over an open-type mining machine operating on the return airway. Proper inspection for gas was not made in either case.

TYPES OF EXPLOSIONS

The classification of mines as gassy or nongassy as given below and in table 1 is based on the evidence as to actual presence of gas in appreciable amounts before the explosion. The Kentucky mining law provides for the

grouping of mines under these headings, and mines are so classified by the State Department, but since the division involves a vague rating as "dusty" and does not conform to U. S. Bureau of Mines recommended standards, the classification as expressed in Mine Safety Board Decision 3 has been followed in rating the mine as gassy or nongassy in the following tabulation:

Types of explosions

Known condition of mine	Explosions of			Total
	Gas	Dust	Gas and dust	
Nongassy	1	23	0	29
Gassy	22	11	11	44
Explosions	23	39	11	73
Fatalities	41	137	166	344

The 73 explosions listed in table 1 are summarized in the above tabulation to show the number of gas, dust, or gas-and-dust explosions in nongassy and gassy mines and the number of fatalities resulting from explosions in each type of mine. Several ignitions occurred in mines not previously known to give off explosive gas; in fact, most of those in early years listed in table 2 were in this class. Several explosions were initiated by gas in mines which were not rated as gassy by the operators or the State. The U. S. Bureau of Mines considers all coal mines potentially gassy; nevertheless, it must be recognized that numerous mines in Kentucky are essentially nongassy, particularly those in the high ridge districts.

TABLE 7. - Fatalities and number of men possibly involved
in Kentucky explosions

Mine rating	Explosions	Gas			Dust			Gas and dust		
		Cases	Fatalities	Men in area	Cases	Fatalities	Men in area	Cases	Fatalities	Men in area
Nongassy	Local.	1	4	10	7	4	85	-	-	-
Do.	Violent.	-	-	-	21	96	164	-	-	-
Gassy	Local.	19	24	1,587	4	5	121	5	32	360
Do.	Violent.	3	13	13	7	32	37	6	134	636

The above table indicates the large number of fatalities due to gas-and-dust explosions in gassy mines and the smaller number of fatalities due to dust explosions in which gas was not involved. Explosions of gas and dust were in numerous cases initiated by ignition of gas accumulations during the working shift when full crews were in the mines; for this reason the loss of life was heavy when the explosions were propagated by dust throughout a considerable part of the workings. Dust explosions that were more or less violent, and the majority were of this nature, occurred more often when only

shot firers or a few other employees were in the mine; in these cases few escaped. Had it not been for regulations, imposed in certain mines, prohibiting shooting off the solid during the working shift, the loss of life would probably have been 500 or 1,000 instead of 128 in these few cases. However, it is unthinkable that such disregard of human life would have been countenanced. The explosions involving gas alone were nearly all localized, since where any amount of violence was developed dust was ignited, bringing the occurrence under the gas and dust division.

PREVENTION OF EXPLOSIONS IN KENTUCKY COAL MINES

It is now fairly well-established that all mine explosions are preventable. The necessary knowledge of the causation of mine explosions and means of absolute prevention to insure protection from them have been available for at least 15 years. Recommendations were made in reports on the various explosions by the State Department of Mines and by the United States Bureau of Mines representatives to prevent recurrence of similar explosions; these recommendations represented the best accepted safety practices at the time they were made. The number of times such recommendations were given is shown in table 8.

Table 3. - Recommendations for preventing Kentucky explosions
made at the time of the explosions

Recommendations	Times made
Cutting or mining of coal instead of solid shooting	38
Adoption of a proper system of ventilation or improvement of ventilation	25
Use of safer blasting practices (abandonment of the use of overcharged or straight holes, coal-dust stemming, etc.) ..	24
Control of dust (watering, dust removal, rock-dusting, water on cutter bar)	22
Installation of permissible (closed) lights	22
Provision of effective inspection	18
Adoption of permissible explosives and blasting methods	18
Provision of adequate supervision	11
Adoption of safer methods of storing and handling black blasting powder	2
Ventilation of mine or parts of it before entering	2
Sealing of unventilated areas	1
Prohibition of smoking	1
Use of permissible electric equipment only	1
Provision of borcholes to release gas from sealed or confined workings	1

Mined coal v. solid shooting.— The following tabulation shows the slow progress from solid shooting to machine mining in many of the mines; little real pick mining was done at any time, and considerable solid shooting is practiced at present.

TABLE 9. - Proportion of machine-mined coal to total production in Kentucky, percent

Year	West district	Southeast district	Northeast district	State
1909	80	27	47	58
1910	74	36	66	62
1911	79	32	74	63
1920	-	-	-	78
1924	-	-	-	87
1928	-	-	-	92
1931	-	-	-	91

The recommendation to cut the coal was made in 38 of the 73 reports prepared in connection with explosions in Kentucky coal mines.

Explosions due primarily to solid shooting were numerous and of more or less regular occurrence until about 1923; several have occurred since that date--2 in 1923, 1 in 1924, 2 in 1925, 1 in 1926, 1 in 1929, and 1 in 1932. All explosions were in mines in which the coal was supposed to be cut but in which more or less solid shooting, supposedly as an exception, was permitted. The tendency is toward total elimination of this practice, and the exceptions should be reduced to zero.

Ventilation.— In 1864, when the first State mine inspector assumed office, most of the mines depended on natural ventilation or on small fire baskets to produce limited circulation. For years he urged better means of ventilation. Furnaces were next installed, and they were numerous as late as 1912. A fan installation was described in 1891, and from that time on fans were advocated to replace furnaces. The virtually nongassy condition of many mines with outcrops on high ridges and numerous openings to the surface has had the effect of minimizing the importance of ventilation and inspection over much of the State.

The recommendation to improve ventilation was made in 25 instances in the reports prepared after explosions; many of these were cases in which lack of adequate circulation of the air allowed smoke and fumes, as well as very finely divided dust, to collect to such an extent after heavy blasting that ignition from a "windy" or blown-out shot occurred. Several such ignitions are also recorded in table 2, giving data on some explosions in which no fatalities occurred; the recommendation to improve ventilation was made in nearly every instance where gas was involved, and in some in which dust was the active factor.

An adequately controlled ventilating system providing for a sufficient quantity of fresh air to be conducted to and to render harmless all gases at every working face is of primary importance in every mine from an explosion and health standpoint.

Safer blasting.- Safer blasting includes recommendations dealing with proper drilling and placing of holes, limiting the amount of black blasting powder put in any hole and the number of holes shot at a time, using clay stemming in place of coal or other combustible dust, and shooting at prescribed times. One of several of these hazards contributed to an explosion in 24 instances. These occurred in the period before permissible explosives were generally known and available; after that time equivalent recommendations are grouped under "permissible blasting".

Permissible explosives and blasting.- The recommendation for use of permissible explosives and recognized safe blasting methods was made in 18 instances; in all probability at least 18 explosions in Kentucky would have been avoided through the use of available safe, effective types of explosives. The following tabulation shows the relative percentages of the three chief types of explosives used in the coal mines of Kentucky at various periods:

TABLE 1C. - Explosives used in Kentucky coal mines

Year	Black blasting powder	Dynamite	Permissible explosives
1911	Nearly all.	-	Some in northeast field and small amount in western field.
1924	83	?	17
1925	82	1	17
1926	82	2	16
1927	76	1	23
1931	68	3	29

In this tabulation all black blasting powder, whether in granular, pellet, or stick form, is grouped together. Until recent years nearly all black blasting powder was in granular form but has been largely replaced by pellet and stick varieties. From an explosion standpoint the risks are essentially the same. Dynamite is used virtually entirely in shooting top rock; in this it constitutes a grave hazard and its use is entirely unnecessary, as types of permissible explosives are available which do this work effectively and safely.

At the beginning of 1933 the State Mining Department renewed its efforts to induce operators to use permissible explosives, with a gratifying response from several progressive companies; by all means the use should be universal in Kentucky coal mines.

Dynamite has been charged with initiating only one explosion, in which an open charge was exploded on a large piece of rock. The hazards of black blasting powder and dynamite are generally well-known; and, although the effectiveness and safety of permissible explosives are established and can be demonstrated, nonpermissible explosives are widely and persistently used, chiefly on account of an inherent reluctance to change from established practices.

Dust control.— In 1932 there were 18 explosions in bituminous mines in the United States. In four of these mines rock-dust localized the explosion and saved 100 to 300 lives. In two other partly protected mines the explosion spread to untreated workings and claimed the lives of 15 or 20 men therein.

Over the period of 49-1/2 years, January 1, 1884 to June 30, 1933, in 73 Kentucky explosions dust control was advised as a major preventive measure 22 times, indicating that in at least that number the initial explosion very probably was propagated by dust, with resulting destruction and loss of life. In almost every other instance the explosion might have been propagated by coal dust so that some means of protection against this hazard should have been provided. The reasons for nonpropagation of flame were chiefly weakness of the starting force and rapid release of pressure into nearby openings although dampness and incombustible material naturally present in the dust aided in at least eight instances.

Rock-dust stopped explosions in two mines. One explosion was confined to a non-rock-dusted entry, in which 17 men were killed; a total of 180 men were employed underground, and it is believed that many if not all of 103 lives were saved in this mine through the effectiveness of rock-dust. Another explosion in the same district was arrested by rock-dust; 15 men were killed, and 30 men escaped, undoubtedly due to the efficacy of the rock-dust. Coal-dust explosibility control through application of rock-dust was responsible in these two instances for saving possibly 193 lives. Natural rock-dusting--that is, dust high in incombustible material--was credited with stopping another explosion in which 3 lives were lost and 6 men escaped.

Rock-dust, to be effective, must be so maintained that the incombustible content of the dust exceeds the minimum necessary to prevent propagation of an explosion. The fine dust of the No. 9 or Keokee bed requires at least 76 percent incombustible matter to make it nonexplosive. Failure to extend and maintain rock-dusting over a period of approximately one year resulted in a relatively large number of fatalities in an explosion in one Kentucky mine; an explosion resulted in the loss of 23 lives in a mine in which 37 men were underground, and although the main entry had been rock-dusted about a year before the explosion the active workings had not been rock-dusted, and the explosion extended throughout the active workings.

In 1895 Chief Mine Inspector C. J. Norwood included in his annual report a review and explanation of studies that proved the explosive properties of coal dust. For years the only known methods of reducing this hazard were removal of large accumulations of dust and sprinkling, combined with shooting after the shift was out of the mine.

The use of rock-dust was actively advised in the State by engineers of the United States Bureau of Mines about 1920, and certain companies in the western field experimented and adopted its use at that time. Experience has shown the need of such protection in all parts of the State, and recently marked interest has been taken in the explosion-prevention method. Several mines have been rock-dusted, and operators of other mines are giving the matter serious consideration.

Permissible (closed) lights.- Twenty-two mine explosions in Kentucky were caused by the use of open-flame miners' lights in gassy mines after practical electric cap lamps had been in use in some States for several years and their use generally advocated in all coal mines and especially in gassy mines. Fifteen ignitions listed in table 2 were caused by open lights in gassy mines. In the years before practical electric cap lamps were developed the use of any one of several makes of flame safety lamps instead of open flame lights when unventilated and uninspected workings were entered would have prevented several disastrous explosions causing nonfatal injuries.

Approximately 137 electric cap lamps were in use in the State of Kentucky in 1918 and about 1,966 in 1928. The number in use in July, 1933, is reliably estimated to be about 5,300. In 1928 there were about 53,600 men in the mines, and there are about 40,000 in 1933.

Present-day electric cap lamps of approved types are more economical, more efficient, and more safe than open-flame miners' lamps; the Bureau of Mines recommends the exclusive use of closed lights in all coal mines, as experience has demonstrated not only in Kentucky but also in virtually every other coal-producing State in the Union that open-flame lights are hazardous.

Inspection.- Eighteen of the 73 explosions were charged to failure, through lack or neglect of inspection, to find and remove gas accumulations before men were permitted to enter, or in other instances to test for gas before shooting or putting power on open-type equipment.

Not only should fireboss inspections be made for gas and close attention be given to the ventilating system to prevent gas from accumulating, but an inspector should also regularly survey dust-control methods, such as watering, cleaning dust from roadways, and rock-dusting; should keep a close check on electrical equipment to see that arcing and sparking are reduced to a minimum, especially at or near working faces; and should see that all open-flame lamps and smoking are barred from all coal mines whether gassy or nongassy.

Supervision.- Direct responsibility for explosions was charged to lack of adequate supervision in 11 reports on these disasters. In a majority of all the cases inadequate supervision was held to be a factor, and recommendations were made that this function be more effectively exercised in connection with various phases of operation.

In 1914 a new mining law in Kentucky provided that the mine foreman should devote the whole of his time to his duties, defined then, including the noting of all dangers and the taking of proper precautions promptly, and made him responsible. The responsibility of those in charge for the acts of employees cannot be evaded, hence supervision should be intelligent, rigid, and continuous.

Miscellaneous recommendations.— Six other recommendations were made as corollaries to the preceding main recommendations; these included blasting, ventilation, sealing of abandoned or unventilated areas, smoking, use of permissible electrical equipment, and provision of boreholes to release gas in sealed workings.

TABLE 11. — Number of explosions and fatalities from them in Kentucky mines, January 1, 1884 - June 30, 1933

Years	Explosions	Fatalities	Average fatalities per explosion
1884-1890 ^{1/}	2	6	3.0
1891-1900	4	7	1.75
1901-1910	20	83	4.15
1911-1920	16	84	5.25
1921-1930	28	135	4.82
1931-June 30, 1933 ^{2/}	3	29	9.67
Total	73	244	3.34

^{1/} 7 years only.

^{2/} 2-1/2 years only.

The explosions and fatalities from explosions have been arranged by periods (and chiefly by decades) from January 1, 1884, to June 30, 1933. It may be noted that the number of explosions and fatalities increased with the development of the industry in the State to about 1930. Table 1 shows that although there had been two or more fatal explosions in Kentucky coal mines annually for many years up to 1928, there has been but one annually from 1929 to 1933, inclusive. The severity is apparently on the increase, although the 2-1/2-year period, January 1, 1931 to June 30, 1933, is too short to use in making generalized comparisons.

RECOMMENDATIONS

The United States Bureau of Mines, through its Mine Safety Board, issues "decisions" on safety methods and practices. Such decisions as relate to explosion-prevention methods are as follows:

Decision 1.— The Bureau of Mines recommends:

1. In all coal mines the portable lamps for illumination be permissible, portable, electric mine lamps; and also

2. In places where fire damp or black damp is liable to be encountered, a permissible magnetically-locked, flame safety lamp for gas detection, or equivalent permissible device, be supplied to at least one experienced employee in each such place; and

3. Any employee before being supplied with a permissible flame safety lamp be examined by a competent official of the mine to assure the man's ability to detect gas; and

4. All coal mines whether classed as nongassy or gassy in any part, be supplied with magnetically locked, permissible, flame safety lamps, properly maintained and in sufficient number for all inspection purposes.

Decision 2.- In the interests of safety the Bureau of Mines recommends that for blasting in coal mines, permissible explosives, fired electrically, be exclusively used; and that as an aid to blasting, all coal which is feasible to cut, should be cut or sheared.

Decision 4.- In the interest of safety, the Bureau of Mines, Department of Commerce, recommends that auxiliary fans or blowers should not be used in coal mines as a substitute for methods of regular and continuous coursing of the air to every face of the mine.

Decision 5.- To prevent the propagation of mine explosion, the Bureau of Mines, Department of Commerce, recommends rock-dusting all coal mines, except anthracite mines, in every part, whether in damp or dry condition. It also recommends that rock-dust barriers be used to sectionalize the mine as additional defense; but these should not be regarded as a substitute for generalized rock-dusting.

Decision 6.- In the interest of safety, the Bureau of Mines, Department of Commerce, recommends that in coal mines all entries, rooms, panels, or sections that cannot be kept well ventilated throughout or cannot be inspected regularly and thoroughly, or that are not being used for coursing the air, travel, haulage, or the extraction of coal, be sealed by strong fireproof stoppings.

Decision 7.- In the interest of safety, the Bureau of Mines, Department of Commerce, recommends:

1. That the main intake and main return air currents in mines be in separate shafts, slopes, or drifts.

2. That the main intake shaft lining be of fireproof construction, and there be a minimum amount of inflammable material in or adjacent to the shaft.

Decision 9.- The Bureau of Mines, Department of Commerce, recommends in coal-mine ventilation practice the following specifications as to unit quantity and quality of air:

1. The quantity in cubic feet of pure intake air flowing per minute in any ventilating split shall be at least equal to 100 times the number of men in that split.
2. The quantity of air entering each unsealed place shall be at least 200 cubic feet per minute and as much more as may be necessary to properly dilute and carry away inflammable or harmful gases which may be present.
3. The air shall be made to circulate continuously to the face in every unsealed place into which an appreciable amount of methane enters.
4. The air in any unsealed place shall be considered unfit for men if it shall be found to contain less than 19 percent oxygen (dry basis), more than 1 percent carbon dioxide, or a harmful amount of poisonous gas.
5. If the air in any unsealed place, when sampled or tested in any part of that place not nearer than 4 feet from the face and 10 inches from the roof, shall be found to contain:
 - (a) More than 1-1/2 percent of inflammable gas, the place shall be considered to be in hazardous condition and require improved ventilation, and
 - (b) If more than 2-1/2 percent of inflammable gas is found, the place shall be considered dangerous, and only men who have been officially designated to improve the ventilation and are properly protected shall remain in or enter said place.
6. If the air in the split which ventilates any group of workings contains more than 1-1/2 percent of inflammable gas, these workings shall be considered to be in a dangerous condition and only men who have been officially designated to improve the ventilation and are properly protected shall remain in or enter said workings.

Decision 11.— In the interest of safety, the Bureau of Mines, Department of Commerce, recommends that in coal mines, haulage and (or) hoisting be kept in intake air as far as possible.

Decision 12.— The Bureau of Mines, Department of Commerce, extending Mine Safety Decision No. 2, recommends that for blasting either coal or rock in coal mines, permissible explosives or equivalent permissible device be used exclusively, and in addition recommends that in blasting

1. Each charge shall be in a hole properly drilled and stemmed with incombustible material.

2. Each shot shall be fired separately by a permissible single-shot blasting unit, using an electric detonator or igniting equivalent of a kind specified by the Bureau for the particular permissible explosive or permissible blasting device.

3. Before and following each shot in gassy and slightly gassy coal mines, examination for gas shall be made with a permissible flame safety lamp or permissible equivalent and

4. If more than 1-1/2 percent of inflammable gas is found, in the quantity and by the method specified in Mine Safety Decision 9, the place shall be considered to be in a hazardous condition and before another shot is fired the gas shall be reduced by ventilation below the percentage and quantity specified in Decision 9.

5. Each shot employing explosives shall be prepared and fired by or under the immediate supervision of a man having a state certificate as a mine examiner, fire boss, or foreman; and whenever conditions permit all other men than those authorized to prepared and fire shots shall be out of the mine when shot firing with explosives is being done.

Decision 13.— The Bureau of Mines, Department of Commerce, recommends that when electricity is used in coal mines rated as gassy, or whenever in any mine the atmosphere may become gassy:

1. Electrical equipment shall be permissible.

2. Nonpermissible electrical equipment shall be used only in pure intake air.

3. Electrical power shall be cut off whenever the air in the workings is in a dangerous condition, due to inflammable gas.

Decision 15.— In the interest of safety in coal mining, the Bureau of Mines, Department of Commerce, recommends that, to lessen the coal-dust explosion hazard:

1. Machine coal cuttings be wet as the cutting is being done.

2. The coal face, and the working place 40 feet therefrom, shall be kept free of coal-dust by the use of water.

3. The top of loaded cars in the working place shall be wet.

Decision 16.— In the interest of safety in coal mining, the Bureau of Mines, Department of Commerce, recommends that:

1. Machine cuttings be removed from the cut.

2. If the machine cuttings are of a character which would contribute to a dust explosion, they shall be sent out of the mine.

Decision 17.- To lessen the formation and distribution of coal dust in haulage ways, the Bureau of Mines, Department of Commerce, recommends that in bituminous and lignite coal mines:

1. The mine cars should be constructed and maintained dust-tight.
2. The coal should be so loaded that it will not shake off in haulage.
3. The cars and loads should be so sprayed as to prevent dust being distributed along the haulage ways.

Decision 18.- In the interest of safety in coal mining, the United States Bureau of Mines recommends that:

1. The foreman regularly in charge of underground operations and also any person who, in the absence of the foreman, may be placed in temporary charge should each have a certificate of competency from the State to act as mine foreman.

2. The superintendent or person in responsible charge of the mine, to whom the mine foreman reports, should have a certificate of competency from the State which should be issued upon a showing of underground experience for a period of time as long as that required for a foreman's certificate and upon passing an examination including all technical questions asked in the examination required of foremen.

3. These certificates should expire after some stated period of time, such as 5 years, and should be renewed only after the applicant has again passed the examination required by the State.

Decision 20.- In the interest of safety in underground mining, the United States Bureau of Mines recommends that while driving tunnels or drifts and sinking or raising shafts or slopes, and also in their operation, there should be an adequate ventilating current wherever men work or travel.

CONCLUSIONS

More than half of the coal-mine explosions in Kentucky mines have been caused by improper blasting. The coal operators finally appear to have realized that changes should be made in blasting practices, and a definite trend has developed toward the use of permissible explosives and blasting supplies. Helpful results undoubtedly will follow widespread adoption of these safer explosives and more reasonable blasting practices.

Open-flame lamps have caused 37 percent of Kentucky coal-mine explosions, and realization of the hazards of this type of lamp by the mine owners is indicated by the fact that 2.3 times as many permissible electric cap lamps were in use early in 1933 as in 1928. Unquestionably open lights should be superseded in Kentucky coal mines by the far safer and more efficient closed lights now readily available.

The 73 explosions in Kentucky mines listed in table 1 represent a total daily average production of 41,195 tons of coal. The total average underground employment represented is 7,988, and of this number 3,127 men were reported to have been in the mine at the time of the explosions. The total number killed was 344 or approximately 11 percent of the men exposed to the explosions; 123, or about 4 percent of those underground, were injured. In other words, nearly 15 percent of the men underground were affected by the explosions. The number (159) that escaped from the mine or the area affected by the explosion represents slightly more than 5 percent of the total number underground. The relatively high mortality rate (55 percent) of the 626 persons in the affected area, consisting of 344 killed, 123 injured, and only 159 uninjured, emphasizes the necessity for adopting dust-control methods. Only about 25 percent of those in the explosion areas escaped; if all mines listed in table 1 had been adequately rock-dusted the number of fatalities in all probability would have been very remarkably reduced.

During the past 5 years an earnest effort has been made to eliminate coal-mine explosions in Kentucky. If the recommendations of the State Department of Mines and the United States Bureau of Mines are adopted generally, especially those for substituting permissible explosives and blasting supplies for nonpermissible explosives and fuse, permissible lamps for open-flame lamps, and improved methods of ventilation, the groundwork already laid undoubtedly will form a permanent foundation for preventing coal-mine explosions, with their attendant unnecessary loss of both property and human life.

UNITED STATES BUREAU OF MINES
SCOTT TURNER, DIRECTOR

INFORMATION CIRCULAR

THE EXPERIMENTAL MINE OF THE UNITED STATES BUREAU OF MINES



BY

G. S. RICE, H. P. GREENWALD, AND H. C. HOWARTH

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UNITED STATES BUREAU OF MINES

THE EXPERIMENTAL MINE OF THE UNITED STATES BUREAU OF MINES¹

By G. S. Rice,² H. P. Greenwald,³ and H. C. Howarth⁴

INTRODUCTION

The Experimental Mine of the United States Bureau of Mines is a unique coal mine that serves as a large-scale testing laboratory. It is used primarily for testing the explosibility of coal-dust and gases but is also the workshop for many other kinds of research work of which the chief object is to increase safety in mining.

The mine is situated in Allegheny County, Pa., about 13 miles southwest of Pittsburgh among the high hills of the west bank of the Monongahela River, at the northeastern edge of the gas-coal district of the Pittsburgh coal bed which extends south through the Connells-ville coking-coal district into West Virginia. The Pittsburgh bed, of the Upper Carboniferous age, is one of the most extensive continuous coal beds in the United States, if not in the world. Throughout large areas it shows remarkable uniformity of thickness, levelness, and character of coal, which is a coking coal of high-grade bituminous rank. Its thickness in the northern part is 5 to 5½ feet and in the central and southern part is 7 to 9 feet.

The Experimental Mine tract, containing 77 acres of surface and 38 acres of coal about 5 feet thick, lies in the spur of a hill, and the coal outcrops around this spur. On the south and southwest the area is bounded by old workings of an adjacent mine from which the pillars have been extracted, but it is separated from these workings by a continuous barrier pillar.

Adjacent to the Experimental Mine openings is the Explosives Testing Station of the Bureau of Mines. The power plant of the mine also serves that station.

The mine was developed in 1910 from the outcrop of the coal bed primarily for coal-dust and coal-dust and firedamp testing. The problems studied may be grouped as follows:

Relative explosibility of coal-dusts of different kinds, their relative ignitability in air and manner of explosion propagation.

Coal-dust explosion limitation and prevention methods.

Firedamp diffusion and explosion processes.

Ventilation methods in mines and vehicular tunnels.

Fire extinguishment apparatus, and other agencies.

Testing of explosives as concerns practical effects.

Strength of mine stoppings of different construction to resist explosions.

Compressibility and bearing strength of coal in place horizontally as a buttress and vertically as a pillar.

Roof movements and testing of roof pressures.

Roof protection methods such as guniting.

¹ The Bureau of Mines will welcome reprinting of this paper, provided the following footnote acknowledgment is used:

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² George S. Rice, chief mining engineer, U.S. Bureau of Mines.

³ H. P. Greenwald, supervising engineer, Experimental Mine Section, U.S. Bureau of Mines.

⁴ H. C. Howarth, mine superintendent, Experimental Mine, U.S. Bureau of Mines.

Other work at the mine includes instruction of miners and officials in mining hazards, also the training of men in the use of rescue apparatus and testing devices etc., and the demonstration of mining hazards, particularly explosion hazards. Such demonstrations have been observed by many thousand persons.

Details of the problems investigated or under investigation are given later.

CHARACTER OF EXPERIMENTAL MINE PASSAGES

The mine is opened into the outcrop on the south side of a narrow valley by two parallel main entries, called the main entry and main air course, which run southward and are 1,300 feet long. There is a short single side entry, with testing chambers at the head, turned off the main entry to the right (westward) about halfway to its face, and two pairs of side entries, termed "butt entries," turned left (eastward) off the parallel main air course. The entries of the inby pair are 528 feet long and can be extended no further because of shallow cover, as their faces approach the outcrop in the valley next to that from which the mine drifts enter. From the outby entry of this pair seven rooms have been turned.

The entries of the outby pair are newer (1929) and have been driven only about 200 feet from the main air course at the time of writing (May 1933). They are parallel with the axis of the spur and will be approximately 1,200 feet long before reaching as close to the outcrop as advisable in order to leave sufficient roof cover. Two narrow headings have been driven southward off the outby pair of butt entries, for carrying on compressibility-of-coal tests. These headings are so located that they can be widened into rooms if desired.

The mine arrangements, as will be noted in the accompanying plan, are similar to those of the conventional room-and-pillar layout of a small mine in the Pittsburgh district. The entries are 9 to 10 feet wide and 50 feet from center to center; the rooms are 20 feet wide and 44 feet center to center. Crosscuts have been made 100 to 200 feet apart but the crosscuts of the main entries have been sealed by strong reinforced concrete stoppings.

The roof and ribs of the standard explosion-testing zone have been gunited and the floor concreted. This is advisable on several accounts: It prevents dust abraded from the Pittsburgh bed from entering into the explosion tests; it aids complete cleaning, which is especially important when the dust under test is changed in character or when dust-density tests are made; and it protects the roof and ribs from weathering and from breaking down from effects of the explosions. The gunite is put on naturally irregular walls with the object of duplicating practical conditions found in mine passages, which irregularity is a factor in causing turbulence of the air in an explosion, and consequent more uniform mixing with air of the dust raised by the advance air waves that always precede an explosion.

CHARACTER OF ROOF AND OVERBURDEN

The immediate strata over the coal bed consist of "draw slate" or clayey shale normally 6 to 10 inches thick but thickening locally to more than 3 feet, above that an interbedded layer of mixed bony coal and shale about 2 feet thick, and over that bituminous shales weak in strength. The surface consists principally of clay, from the weathering of the shales, and a thin layer of soil. The maximum thickness of cover over the mine is 130 feet at a place about 775 feet from the mouth of the main entry.

Face and butt joint planes are strongly marked in the coal, as is characteristic throughout the Pittsburgh bed. The direction of the main entries and the room is nearly at right angles to the "face joints."

POSITION OF MINE WITH REFERENCE TO SURFACE TOPOGRAPHY
AND OTHER MERITS OF LOCATION

As previously mentioned, the main entrances of the mine are drift openings in the south side of a narrow valley and in a line directly opposed by the steep north bank. When explosions are of violent character the side hill receives the force and flames of the explosion and deflects them upwards, thus preventing, except in the most violent explosions, the air wave effects which are likely to cause damage to dwellings in the vicinity. This is important, as there has been much destruction of windows in the vicinity of surface galleries in Europe where these galleries were in flat country as at Liévin, France, and Altofts, England. Even with this protection, very violent explosions issuing from the Experimental Mine have broken windows at points over a mile distant.

Other factors entered into the selection of the location of the mine in 1910. It was desired to obtain an area of coal bed which would be practically free from methane given off by the coal (which condition was fulfilled by the site chosen on account of unrestricted drainage of gas to outcrops) yet have gas available for testing. The latter condition was obtained by having natural-gas wells and a gas line in the vicinity from which gas is obtained for making tests. Another condition sought was to obtain a coal area where the mine would be virtually self-draining of water. At the location chosen there is a slight dip of the bed toward the entrances so that only a small amount of pumping from depressions is required. With the exception of the faces of the inner pair of left butt entries, the coal bed had been found naturally dry.

HISTORY

A series of great coal-mine explosion disasters occurred a quarter of a century ago which caused World-wide apprehension in the coal industry. These began with the Courrières disaster in France (1906) in which 1,100 men were killed; this was followed by a number of major explosion disasters in England and by many in the United States in 1907, culminating in a great explosion at the Monongah Mine in West Virginia in which 358 men were killed. These happenings naturally caused all coal-mining men much anxiety and led to the appropriation of funds for Federal investigations into the causes of coal-mine explosions and other mine accidents. The investigations were placed under the Technologic Branch of the U.S. Geological Survey, which had been carrying on fuel investigations, and the appropriation was made available July 1, 1908.

Mining engineers were engaged to study the effects of explosion disasters. These investigations were supplemented by chemical investigations of different coals and mine gases, conducted in a laboratory and testing station established on the old Arsenal grounds at Pittsburgh.

At that period the explosibility of firedamp or methane was well known, as was the inflammability of coal-dust, but most mining men considered that it was not possible for coal-dust to propagate flame from a source of ignition by itself, and that the presence of methane was always required. However, this belief was shaken by explosions occurring in so-called nongassy mines; at that period these occurred most frequently at the time of firing shots. Laboratory testing in small models and also testing in short surface galleries had been extensively tried in various countries but with very conflicting results because, we now know, the size and kind of coal-dust, the density of the dust cloud, the amount of inert matter present, the source of ignition, and the release of pressure through openings or joints were factors seldom controlled, neither was their significance understood.

In 1907 the French established a testing gallery not far from the Courrières Mine in the north of France and in 1908 the British established a somewhat similar gallery near the Altofts colliery, England, which colliery in the past had experienced a disastrous explosion. At the beginning of the Federal investigations in this country, engineers were sent (1908) to study the testing then being carried on in France and England. As a result of their report a gallery 100 feet long by 6 feet 4 inches in diameter was installed at Pittsburgh, principally, however, for the testing of safer explosives for use in gassy and dusty coal mines.

In this gallery many demonstration tests showing the explosibility of coal-dust were carried on in the presence of mining men, but it became evident from the comments by these men that they were not convinced that a relatively short gallery could duplicate conditions in a coal mine. Following the establishment of the Bureau of Mines by act of Congress effective July 1, 1910, to take over the mining investigations and fuel testing of the U.S. Geological Survey, the late Dr. Joseph A. Holmes, then director of the Bureau of Mines, decided to obtain a mine for experimental purposes. He instructed the chief mining engineer, George S. Rice, to find a mine, preferably in the Pittsburgh coal bed, as mines in this bed had experienced a large number of explosion disasters.

After examination of a number of abandoned mines which were offered by their owners, none of which was deemed suitable, it was decided to open a new mine to obtain the desired arrangements.

Various tracts of coal were examined with the cooperation of Pittsburgh coal operators, and finally a suitable tract of Pittsburgh coal was selected and leased from the Pittsburgh Coal Co. The "Experimental Mine" was then started in the summer of 1910 by driving entries into the outcrop of the coal bed.

Some years later when the Government investment in the mine development, structures and equipment had become increasingly large, negotiations were begun for purchase of the coal tract and the surface lands needed for use in the mine testing work and the explosives-testing laboratories and equipment which had been moved from Pittsburgh to the site adjacent to the Experimental Mine. In 1924 this purchase was consummated.

DETAILED PROBLEMS STUDIED

The following problems have been studied. Some of them have been completed; others are continuing or may be taken up again when time or financial conditions permit:

Testing of Explosibility of Coal-dusts

The problem of coal-dust explosibility has developed so many ramifications during the years of study and testing devoted to it that its essentials are best grasped by recognizing certain subdivisions. The present summary of testing is divided into three parts:

(A) Tests having to do primarily with the ignition of coal-dust and the initiation of explosions;

(B) Propagation of explosions after initiation; and

(C) General physical and chemical phenomena of explosions.

The foregoing testing applies to different coal-dusts with various characteristics.

(A) Ignition of coal-dust:

1. Initial ignition of coal-dust by different sources:

(a) By explosives, either in blow-out or mud-capped shots. Tests have been made of the ignition of coal-dust by dynamite, which is sometimes used in mud-capped shots to break large rocks as in coal mines. These tests and demon-

strations are conducted on the outside of the mine as well as in one of the portals, and demonstrate the hazard of this method as compared with the use of block holes.

- (b) By the explosion of gas-air mixtures prepared by admitting gas behind a diaphragm erected to confine the mixture to a definite volume.
- (c) By electric arcs.

Tests have been made to determine the minimum electrical arc which will ignite a pre-made cloud of coal-dust; this has special application both at the face of the mine where loading machines are used and on a trolley haulageway in which there may be wreckage of a trip of cars accompanied by short circuiting of a trolley wire, or, of a power line, as occurred in a slope mine in Alabama. Ignition may also occur by short circuiting an electric lighting system, as was claimed to be the cause of a disastrous explosion which originated in intake air close to the mouth of a slope mine in the Pittsburgh district and followed immediately the breaking down of a coal conveyor which ruptured electric lighting wires at the foot of the slope.

- 2. The effect of confinement on ignition - that is, whether the coal-dust is present in narrow places such as headings, or in wide places such as rooms.
- 3. The minimum quantity of coal-dust that can be ignited with production of a self-sustained explosion, and the variation thereof with size of dust, strength of source of ignition, and position of the dust with respect to the perimeter and cross-section of the passageway. The position of the dust determines the ease with which a dust cloud is formed by a given source.

(B) Propagation of coal-dust explosions:

Grading or rating of the relative explosibility of different coal-dusts, or of the same coal-dust under different conditions, has been based on the minimum amount of inert or noncombustible material that must be present to prevent propagation of flame over distances fixed by certain standard test methods that have been developed. Variations in the relative explosibilities so determined have been studied extensively.

- 4. The relative explosibility of coal-dusts prepared from samples brought from different coal-mining districts of the United States, Canada, and Great Britain. Correlation of the results of these tests with the chemical composition of the coals has shown the effect on explosibility of variation in the percentage of volatile matter, ash, and moisture of the coal itself. Other things being equal, the higher the volatile combustible ratio, the more easily in general does the dust ignite and propagate an explosion.
- 5. The reduction in explosibility caused by extraneous moisture as compared with that produced by extraneous dry inert material.
- 6. The increase in relative explosibility caused by the presence of small amounts (usually 1 or 2 percent) of firedamp. In these tests natural gas is piped to the fan and liberated at the intake orifice thereof. It is mixed with and carried by the air current through the mine.
- 7. The effect on explosibility of variation in the relative positions of coal and inert dust prior to ignition - that is, whether the dust is on the floor, ribs, or timbers.
- 8. The effect on explosibility of size (a) of the coal-dust particles and (b) of the inert dust particles.
- 9. The effect on propagation of varying the quantity of dust present per linear foot of passageway or of dust-cloud density per cubic foot of space. In connection with this there have been determinations of the degree of concentration or

density of coal-dust in air most favorable to propagation of flame and, consequently, requiring the greatest proportion of inert materials in a mixture to prevent propagation. Variation in the density giving this optimum effect caused by altering the source of ignition has been studied, as well as the effect of large quantities of dust near the source of ignition, known as superdusting, whereby development of a self-sustained explosion may be prevented under exceptional conditions. Large quantities of dust along the path of an explosion may retard but do not stop it.

10. The effect on explosibility of dust of the degree of wetness of the walls of the passageway and of the humidity and temperature of the mine air.
11. The effect, at the time of ignition, of movement of ventilating currents of different velocities and direction with reference to the point of ignition.
12. The effect on propagation of a coal-dust explosion of turns and curves, changes in cross-section, and of obstacles such as mine cars in the path thereof.
13. The effect of branches from the passageway traversed by a coal-dust explosion giving opportunity for release of pressure ahead of and (or) behind the explosion.
14. The effect of dustless and watered zones of various lengths in the path of a coal-dust explosion.
15. The effect on propagation of an explosion of altering the point at which it is initiated - that is, whether at the face of a mine in a room or heading, in a passageway at a distance from the face, or near the mouth of the mine.

Quantitative determinations have been made on many of the foregoing problems, particularly as regards the effect of chemical composition of the coal, gas in the air current, extraneous moisture, size of dust, and quantity of dust present.

(C) General phenomena of explosions:

The progress of a coal-dust explosion in a mine is attended by a complex series of physical and chemical phenomena. Comprehensive data on some of these have been given by the recording instruments used in tests as follows:

16. The velocity of the shock wave sent out by the source of ignition.
17. The chemical composition of gases ahead of, in, and behind the flame as determined by analysis of samples collected by automatic samplers.
18. The rapidly fluctuating pressures produced by explosions as recorded by manometers placed at different points in the passageways traversed by the flame.
19. The velocity of the flame at different points as measured by the fusion of tin foils in electric circuits and its relation to pressure waves.
20. The extent of the flame as measured by burning of safety matches and by photographic means.
21. The direction of movement of air and burned gases forced through the passageways by the pressures produced during the explosion.

Prevention of Coal-Dust Explosions

Many methods for prevention of coal-dust explosions have been tested, including:

22. Humidifying as by steam.
23. Wetting the coal-dust.
24. Rock-dusting.

Of the foregoing, humidifying the mine air was found to have no practical effect. Wetting was effective if the coal-dust was so wet as to be in a muddy condition, but in practice it was found not possible to wet with sufficient frequency to prevent explosions; this fact

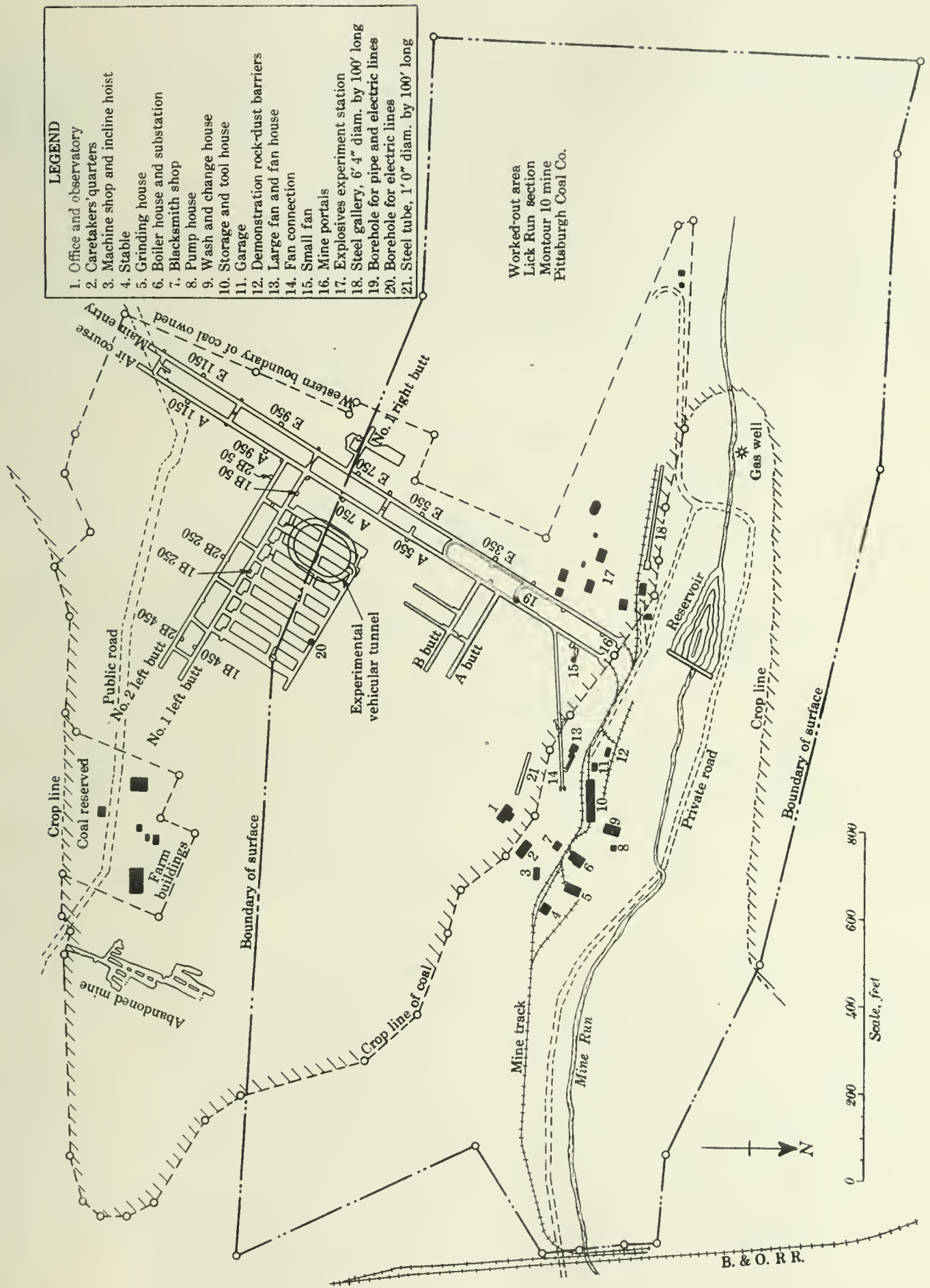


Figure 1.—Plan of Experimental Mine property showing principal surface features and extent of underground workings as of June 1933.

was determined not only in the Experimental Mine but also in commercial mines where the wetting system was employed yet explosion disasters occurred. Rock-dusting was found to be effective and since 1913 has been advocated by the United States Bureau of Mines, to be supplemented by watering at the face of the mine.

Limiting or Stopping Incipient Explosions by "Barriers"

25. "Barriers" of various kinds have been tested, including water barriers, but only "rock-dust barriers" of specific character and with a minimum loading of rock-dust were found to be effective in stopping incipient explosions. The Bureau of Mines has issued a list of barriers of approved character for particular positions but regards them as only supplementary defenses and not as a substitute for rock-dusting.

Firedamp Studies

26. The diffusion of methane as it enters from the face of the mine simulated artificially by the placing of pipes from which gas issues.
27. The effect of ignition of gas (natural gas) on causing ignition of coal-dust:
 - (a) When undiffused,
 - (b) When partly diffused,
 - (c) When thoroughly mixed with air.
28. The length of flame produced by a gas-air mixture when ignited in a heading.

Ventilation Studies and Tests

29. Use of auxiliary blowers and tubes in a heading giving off gas at the face and the increase in concentration of gas in the air in a heading caused by recirculation of air when the intake of the blower is fed partly by the return air issuing from the heading.
30. Measurement of the frictional resistance of a ventilating current caused by:
 - (a) Rough surfaces in a mine passage,
 - (b) Turns,
 - (c) By obstacles, such as cars.
31. Ventilation of vehicular tunnels. An extensive series of tests was carried out in cooperation with the New York - New Jersey Tunnel Commissions, with a view to a proper design for what was subsequently known as the Holland Vehicular Tunnel under the Hudson River. The study was carried out in a specially constructed passageway, elliptical in plan view (as indicated in fig. 1), divided in cross-section into an underduct for intake air, a roadway in which automobiles were run, and an over-duct for return air. This method of ventilating a long tunnel, known as the transverse system of ventilation, was adopted after successful testing in distinction to the previous customary longitudinal system of ventilation. This novel method of tunnel ventilation has been used in various vehicular tunnels in the United States and is to be employed in one which is now under construction in Belgium. The principle of it, by employment of ducts, could be used with great safety in the driving of all long tunnels, such as in those for water supply of cities.
32. Physiological tests of inhaling small percentages of carbon monoxide in the Experimental Mine supplementing laboratory testing.

In addition to the elaborate tests of the movements of the ventilating currents, by the new method of tunnel ventilation, physiological tests were made of the effects of small amounts of carbon monoxide coming from the exhaust of automobiles upon the drivers of the automobiles and others taking part in the tests. As many as 55 persons were employed in a single test.

Mine-fire Extinguishment

From time to time tests have been made of various kinds of fire extinguishers and fire-extinguishing methods. Included among these were:

33. Tests to determine the trajectories of streams of water at a pressure which would permit the handling of the hose and nozzle by one man in a mine passage.
34. Tests of the relative effectiveness of different chemical extinguishing sprays and gases.
35. Rock-dusting by shoveling rock-dust on a mine fire in its initial stage, a method which has proved very successful.

Tests of the Deterioration of Explosives

36. Tests of the deterioration of explosives when stored in mine air are carried on from time to time in certain parts of the mine distant from the entries in which coal-dust explosions are made. These tests are made by the explosives testing section of the Bureau.

Strength of Mine Stoppings

37. An extensive series of mine-stopping tests was conducted in cooperation with the Bureau of Standards. The preliminary tests were carried on in a surface chamber at the Bureau of Standards but the major tests were conducted in the Experimental Mine. This testing was done to furnish information in the administration of regulation of coal mining on the public domain, which calls for stoppings which will resist specific explosion pressures at boundary pillars where openings had been made. The results of these tests determined that a plain concrete stopping acting as a flat arch, when of a certain ratio of thickness to span, would give the most practical form of mine-stopping construction.

Compressibility of Coal

40. As an outgrowth of the tests of the strength of stoppings acting as flat arches and pressing against the coal abutments, testing was conducted on the compressibility of coal in place:
 - (a) Horizontally, which concerns the abutment strength, and
 - (b) Vertically, which has relation to the strength of the coal as a pillar.

A progress report has been issued as Technical Paper 527.

Ground Movement and Subsidence Studies

41. In connection with the extraction of pillar coal along a retreating breakline in the adjoining coal mine, monuments were established and surveys made to determine -

- (a) Whether or not in shallow workings there was a "draw" over the pillar coal, and
- (b) Whether or not there was a "draw" over the barrier pillar adjacent to the Experimental Mine.

The answer was negative under the conditions in both cases, which, however, might not be the case where mine workings were deeper below the surface.

Prevention of Falls by Guniting

- 42. The effect in preventing weak mine roof from weathering and falling is shown by the practical and successful use of guniting for preventing the Experimental Mine entries from caving. This guniting method of protection from oxidation of mine walls was first used and developed in the Experimental Mine in 1913.

World-War Studies

During the World War a number of special studies were carried on at the request of the Chief of Engineers of the Army.

- 43. In the course of testing the use of the French military geophone, geophone was greatly developed by a physicist of the Experimental Mine, and following the war was employed for various purposes, as for example, to detect leaks in compressed-air lines, water mains, etc.
- 44. Studies were made with the improved geophone of the distance at which sounds of working in adjacent places with pick, drill, or machine could be heard through solid strata. The information obtained was primarily for military mining but was later found valuable in certain instances in metal mining in making connections between mine headings, as from shafts or winzes, which were so tortuous that precise instrumental surveys were difficult.

Helium Storage

- 45. The storage of helium underground under high pressure for military purposes in disused or specially constructed mine chambers was considered a possibility, and, at the request of the War and Navy Departments, tests were made in the Experimental Mine. However, later it was decided that storage on the surface in high-pressure steel containers was preferable.

Safety Training and Demonstrations

- 46. One of the most practical educational uses of the Experimental Mine has been as a training place for the candidates for State inspectorships and mine foremen's certificates. The mine passages and arrangements are such that temporary defects in ventilation and other mine safety features can be made. The mine is then examined by candidates, who are requested to make notations of those defects they observe, including such data as results of tests for gas.

Demonstrations

- 47. Demonstrations of the explosibility of coal-dust and the beneficial effects of rock-dusting and other features involved in safety in mines are made to groups

of mining men, as for example, to those having coal-dust under test, or when they are attending meetings of institutes or safety societies in Pittsburgh. Among other demonstrations usually a large explosion test is shown; sometimes the explosion is arranged so as to burst out of the two entrances to the mine with great violence, projecting cars and timber across the valley. Such explosion demonstrations have been witnessed by thousands of mining men. The effect of rock-dusting is usually shown by rock-dusting one or more parts of the mine, through which, in consequence, the explosion does not travel, or the effectiveness of approved types of rock-dust barriers may be demonstrated. At times of such demonstrations other mining hazards are shown such as "mud capping" shots of explosive and direct ignition of coal-dust in air by electric arcs or by an open light.

DESCRIPTION OF PROPERTY, SURFACE BUILDINGS, AND EQUIPMENT

The boundaries of the property and the principal surface features are shown on Figure 1, together with the underground workings and the crop line of the coal. The surface, of which there is 77 acres, is roughly rectangular in shape with the principal axis in an east-west direction. The northern and southern boundaries are at the tops of hills which enclose the valley of the small stream shown as "Mine Run" on the figure. This stream is dammed just west of the mine portals to form a reservoir of boiler water. A private road, which follows the grade of the stream, joins a highway about 350 feet east of the point where it crosses the right of way of the Baltimore & Ohio Railroad.

The coal bed outcrops along both sides of the valley; that part owned by the Bureau is on the south side and has an irregular crop line, as shown in the figure. The south crop line of the government coal is in the valley south of that of Mine Run. The coal under a group of farm buildings in this valley has been reserved. Just east of these buildings is a small abandoned country mine which has been surveyed as far as still open, but of whose history little is known. It has not been worked for more than 40 years.

The mine buildings are grouped to the east of the portals; they have been numbered on figure 1 and their designations tabulated. At the southeast corner of the group is the office and observatory (1). In the latter are housed all instruments used in recording phenomena of coal-dust explosions that are not installed in the mine; also all control and operation during an explosion test are centered there. Immediately below the office are the caretaker's quarters (2) and below these is a small machine shop (3) in which is housed a hoist for handling mine cars on the incline between the mine and railroad levels. Only animal haulage is used elsewhere and the most easterly building (4) is a stable.

Turning westward one encounters next the grinding house (5) in which are crushing and pulverizing equipment and storage bins for dust. All dust mixtures used in explosion tests are prepared in this building. The next building (6) is the boiler house and substation. Steam is used only for heat in cold weather and overhead lines run to all the important buildings, including the explosives testing plant of the Bureau situated west of the mine. Electric power for both the mine and the explosives plant are obtained from a public utility company's, 2,200-volt line which traverses the property as shown; a short branch terminates in a bank of transformers back of the substation. From this there is available for operation and experimental purposes electric currents of 110 and 220 volts d.c., 110 and 220 volts single-phase a.c., and 440 volt three-phase a.c.

All lines used in explosion testing in the mine pass through the observatory where the controls are located, then go to a set of disconnecting switches on the surface near the

portals and finally enter the mine through a borehole (19) near the junction of the aircourse and the first crosscut to the main entry. Power can also be taken into the mine through a borehole (20) at the face of No. 6 room off No. 1 left butt entry. The substation also houses an air compressor from which pipe lines run to the grinding house, observatory, and into the mine, where compressed air is used in the test work, principally in cleaning the explosion test zone.

Immediately south of the boiler house is a blacksmith shop (7). Northwest of it is a small pump house (8) built over a drilled well from which the domestic water supply is obtained. A second pump takes water for general purposes from the surface reservoir which is formed by damming up "Mine Run." Separate tanks sunk in the ground on the hill are used for storing water obtained from these two sources of supply.

Adjacent to the pump house is a wash and change house (9) and above it is a storage and tool house (10). At the west end of this building is a garage (11) and a set of rock-dust barriers (12) erected on the surface for demonstration purposes. Across the mine track is the large main mine-ventilating fan (13) with arrangements for reversing the ventilating current. The fan is connected by a 6-foot diameter steel tube (14) to a branch entrance of the air course. This arrangement allows only a fraction of the force of an explosion in the mine to proceed toward the fan, and there are a number of additional pressure releases on the steel tube and its connection to the fan casing. A small auxiliary fan (15) is connected to the air course through a shallow shaft not far from the portal. Ample pressure-releases are provided for this fan also.

Massive concrete portals of the mine (16) are at the mouths of the main entries. The underground layout has been discussed previously. The portion of the property west of the mine portals is occupied by the Explosives Experiment Station (17), a section of the explosives division of the Bureau. All large-scale tests of explosives are conducted there. One piece of equipment is a steel tube or gallery (18) 6 feet 4 inches in diameter by 100 feet long used primarily in permissibility testing, but available for tests of coal-dust when only small quantities are at hand or there are other reasons for its use in this manner. There is also a tube (21) 1 foot in diameter by 100 feet long installed just west of the observatory for experiments on the propagation of flame in mixtures of natural gas and air.

Passing down the valley of Mine Run is a natural-gas line which collects gas from a number of wells west of the Experimental Mine property. A branch of this line makes available for testing purposes a natural gas of high methane content at both rock and ordinary distributing pressures. Tests have shown that the explosive properties of this natural gas are much the same as those of pure methane, the principal difference being that the limits of explosibility of the natural gas mixed with air are slightly lower.

RECORDING INSTRUMENTS

The nature of the phenomena occurring in a coal-dust explosion can be studied on a large scale only through the use of recording instruments. Some of those at the Experimental Mine were copied from instruments in use at the original French testing station at Liévin and were subsequently modified as required. Others were developed as the work progressed. The classes of records that have been obtained to date are velocity of flame, extent of flame, time-pressure records at different points, composition of gases ahead of, in, and behind the flame, direction of movement of bodies of air and burned gases, and time of movement of different objects such as parts of barriers.

All of these involve the accurate measurement of small intervals of time and chronographs are employed for this purpose. Briefly, these consist of brass drums around which are glued sheets of metallic-surface paper which are covered in turn with camphor smoke. Thirty-

four celluloid pen tips arranged in groups rest on the paper and trace lines in the smoke when the drums are revolved; the pens are mounted on a carriage having a longitudinal movement so that the lines are spirals. The pen tips are attached to the armatures of small electromagnets which when energized impart a movement of about 1 mm. to the tip in a direction at right angles to that of the spiral trace, thus making a short cross-mark which permits measurement of time of the respective movement.

To record any event it is merely necessary to arrange that it will make or break the current in one of the pen circuits. It is possible to record 27 independent and simultaneous events. Normally only one event can be recorded per pen; it is possible to record more than one only when their sequence is invariable and they alternately make and break the circuit. One pen on each chronograph is operated by a tuning fork giving a record in hundredths of seconds from which thousandths are readily estimated. Zero time is taken as the moment at which the explosion is initiated and is obtained by having the source of ignition break the current in one circuit. The normal length of record of an explosion is about 10 seconds; after it is obtained, the papers are suitably marked, the smoke set with shellac solution and the sheets removed for computation.

The chronographs are housed in the observatory. From them leads a 31-wire cable containing a wire for each pen, a common return, a telephone circuit, and a pair of wires reserved solely for firing shots or initiating explosions by other means. This is paralleled by a second 4-wire cable carrying light and power to the instruments placed in the mine. These cables pass above ground to a borehole, as mentioned above, and in the mine are carried in conduits cemented in the ribs to the various stations where instruments are located. Large shelters for instruments are provided at 200-foot intervals throughout the mine; they are indicated on figure 1 by the figures E 350, E 550, A 550, etc. Smaller shelters used only in determining the velocity of flame are at intermediate points.

Velocity of flame is determined through the fusion of small strips of tin foil set in recessed blocks at 100-foot intervals. Fusion of each foil breaks a circuit on the chronograph, and velocities are computed from the time intervals. These foils also serve to determine length of flame to the nearest 100 feet, but a closer measure thereof is needed. This is obtained photographically by the action of the light of the flame on circlets of photographic bromide paper placed in suitably designed receptacles fastened to the roof at 25-foot intervals.

Pressure is measured by means of a specially-designed manometer in which the pressure causes deflection of a steel diaphragm. This diaphragm is connected to a focusing mirror and movement thereof causes a spot of light to travel across a photographic film rotated by a motor. The pressures corresponding to different deflections are known and an auxiliary device makes simultaneous time records on the films and on one of the chronographs in the observatory. It is then possible to translate the records into coordinates of time and pressure and relate them to the travel of the flame recorded by the tin foils.

There is also a tube with lens connected to each manometer so that the light of the flame of the explosion as it passes is recorded directly on the photographic film with the pressure records, thus indicating the duration of this flame on the same time basis.

Differences in pressure between points along the passageways cause violent movement of air and burned gases. The direction of these is determined by the movement of a small vane attached to an instrument so that the vane normally hangs in a neutral position. Movement of the vane in either direction is recorded on the chronographs. It would be desirable to determine the velocity as well as the direction of gas or air movement, but no thoroughly reliable instrument has been devised as yet to supplement the manometer records.

A complex series of chemical reactions occurs in the flame, the oxygen of the air combining with combustible matter of the coal-dust to form gases which are mixed with

products of distillation of the coal-dust. An automatic device has been used for snatching samples of the gases for subsequent analysis. It consists of an evacuated brass bottle of which one end is closed by a valve and the other is sealed with a thin, flat glass plate. A mechanism tripped by an electric current interrupted at a certain distance in by the sampling device, is rotated spirally, the glass is driven against breaker prongs and then, after an interval of one twentieth of a second, against a rubber stopper which traps the sample that has entered. The sample is subsequently transferred through the valve to a glass container and sent to the laboratory for analysis.

SOME PHENOMENA OF EXPLOSIONS

Use of the above instruments in nearly 1,500 explosion tests in the Experimental Mine has given definite knowledge of many of the phenomena that are a part of coal-dust explosions.

The Shock Wave

Firing a shot or a considerable fall of roof in a mine produces a concussion which spreads through the air at the velocity of sound - in fact it is this wave which gives one the sound of the shot or the fall. The velocity of this concussion is about 1,100 feet per second in the Experimental Mine, but is affected by the density of the air and would be different in deep mines or at high elevations. When a shot blows out, the concussion is very violent and has great power to raise dust and form a cloud of it in air. A blow-out shot of 4 pounds of FFF black blasting powder is used in two of the standard test methods employed in the Experimental Mine to initiate a coal-dust explosion. The concussion produced by this shot has been termed the "shock wave." With the cannon in a fixed position the time of its arrival at any given point varies not more than 0.003 second, and it may be that these variations are errors of determination. The pressure of the wave diminishes as it travels and attenuation is particularly marked at points where expansion into branching passageways takes place. It is capable of producing a dense dust cloud in the vicinity of the cannon, but it is doubtful if this power is retained for more than a few hundred feet of travel.

If ignition of coal-dust follows the initial shock wave there is a rapid succession of air waves from the explosive combustion accompanied by a bodily forward movement of the air as a column. These waves and forward movements continue as far as the coal-dust explosion propagates, stirring up the dust ahead of the flame. It is this forward movement with its accompanying waves raising coal-dust that makes spread of a coal-dust explosion possible.

When a coal-dust explosion is initiated by a gas explosion there is no strong initial shock wave unless it is produced by the agency that ignites the gas. Otherwise the first pressure waves are produced by the burning gas; these have all the properties of shock waves and are accompanied by the violent forward movement of the air. When a preformed coal-dust cloud is ignited by an electric arc, the first pressure waves, following the more or less weak concussion which stirs up the coal-dust, come from the combustion and these also have the properties of shock waves.

Flame Velocities

Whereas the initial velocity of a shock wave is 1,100 feet per second or more, depending on the conditions present, the initial flame velocity in a coal-dust explosion will rarely exceed 500 feet per second and may be as low as 100 feet. The flame velocity depends primarily on rate at which combustion can proceed through the dust cloud present, and secondarily

ly on the movement of air carrying the dust cloud along the passageway. If conditions are unfavorable the flame may slow down and die out. The lowest flame velocity ever recorded in the Experimental Mine in a self-sustained dust explosion was 30 feet per second (20 miles per hour, approximately). More commonly the minimum velocity has been between 50 and 100 feet per second. Under favorable conditions the flame accelerates rapidly and velocities of 1,000 to 1,500 feet per second may be attained in 1 second or less. There is some uncertainty as to the maximum velocity attainable in a coal-dust explosion. Velocities in excess of 3,000 feet per second (34 miles per minute) have certainly occurred in the Experimental Mine, and 6,000 feet has been indicated, but the possible experimental errors become so large at such high velocities that considerable uncertainty exists concerning them. At any velocity in excess of 1,200 feet per second the flame can overtake the initial shock wave if a sufficient length of passageway is present, and will then proceed through a dust cloud formed at its front by the pressure of the expanding gases of combustion, projecting its own waves sufficiently ahead to stir up the amount of coal-dust necessary for propagation of the explosion.

Shape of Flame

The flame of a coal-dust explosion in a mine passageway is pointed with the point somewhere near the center of the passageway. Doubtless the actual shape varies considerably from moment to moment, but the general pointed shape is retained. In a weak slow explosion the point is sharper and combustion may not extend to the walls. Some experiments have indicated that this point may be as much as 100 feet long and combustion may be taking place over a length of 300 feet or more at any one instant. As the flame velocity increases, the point of the flame will become blunter and the total length of flame will be shorter because of the more rapid rate of combustion laterally as well as forward. In this connection it should be remembered that the total amount of combustion is limited by the quantity of oxygen present, but possible variations in this quantity are small compared with those of flame velocity and rate of combustion.

Pressures Produced by Coal-dust Explosions

The pressures produced by burning coal-dust at the time of initiation of an explosion are low and usually of the order of 3 to 5 pounds per square inch unless the source of ignition has been exceptionally strong. Should conditions be unfavorable, no higher pressures will be produced during the course of the explosion. On the other hand, with favorable conditions pressure will rise rapidly and reach 25 to 30 pounds per square inch in less than 1 second. Pressures of 60 to 70 pounds are common when good clouds of relatively pure coal-dust are formed.

Theoretically the maximum pressure obtainable with an atmosphere pressure is 142 pounds per square inch, but as the explosion develops, precompression of air occurs ahead which may be several atmospheres; then if there is sufficient coal-dust present, which there nearly always is in a coal mine, the pressure of the explosion is correspondingly increased and extreme violence may result. It is characteristic of explosions in dusty coal mines unprotected by rock dusting that the violence of the explosion increases as it travels. On the other hand, when by rock dusting or for other reasons the flame is extinguished, the pressure rapidly drops.

The highest pressure recorded in Experimental Mine tests was 127 pounds obtained with pure dust of a low-volatile bituminous coal. Higher pressures have been present, as manometers have been wrecked by them. As a rule, the higher the pressure, the less its duration.

Pressures of 50 pounds per square inch or more usually last only a few hundredths of a second and approximate a hammer blow.

Chemical Reactions Occurring in a Coal-Dust Explosion

Some knowledge of the chemical reactions taking place in the flame of a coal-dust explosion has been obtained from the composition of samples of gas taken by the automatic samplers described above. These samples were collected at a point 350 feet from the origin of an explosion, and the flame was well developed before the sampling point was reached.

If one can imagine himself stationed at the point where the samples were collected, with the flame approaching him along the passageway, the first question is whether radiant heat of the flame is having an effect in advance of that flame. No positive evidence of such an effect has been obtained. If it is present the distance over which it is effective is too short to permit detection; a dense dust cloud likely absorbs or screens the heat.

Simultaneous samples collected close to the rib and at a point 27 inches therefrom have shown that vigorous combustion may be in progress at the latter point while little or none is occurring at the former. This is due to the pointed shape of the flame described above. In a weak explosion a thin layer of relatively pure air may remain close to the rib, probably embraced in a dense layer of the suspended dust, until after the flame has passed, while in the center of the entry the oxygen content of the air has been reduced to not over 2 or 3 percent with an accompanying production of 10 or 12 percent carbon dioxide and appreciable quantities of carbon monoxide and distillation products from the coal-dust.

In a rapid and violent explosion of dry dust, combustion is much more complete and the oxygen content of the entire body of the air may be reduced below 1 percent. Carbon monoxide will be present in quantities of 4 percent or more, but the quantity of carbon dioxide is reduced at the same time. This indicates that part of the carbon monoxide is produced by reaction of carbon dioxide with hot carbon after the primary combustion has taken place. The effect of higher temperatures or more prolonged heating is shown by the presence of larger quantities of methane and hydrogen distilled from the coal. Appreciable quantities of ethane and unsaturated hydrocarbons are found at times.

When the coal-dust taking part in an explosion is damp or wet, a new chemical reaction is possible; namely, that between steam and hot carbon, commonly called the water-gas reaction. The hot carbon reduces the steam, forming large volumes of carbon monoxide and hydrogen. The extent to which this reaction can take place is shown by one sample taken in the flame of a strong explosion of wetted coal-dust. This sample contained 0.8 percent oxygen, 11.7 percent carbon monoxide, 6.2 percent hydrogen, 4.2 percent methane and ethane, and 8.4 percent carbon dioxide. The methane and ethane came entirely from the coal-dust, as there was no gas in the air current prior to the explosion.

Chemical reactions practically cease after flame has passed, where upon the gases cool. Any change in composition is then caused by dilution with fresh air from points not reached by the explosion, or by absorption of some of the gases into the coal-dust or coal walls. In any case the remaining gases will be irrespirable until full ventilation is restored.

Coking Phenomena

Coke found after coal-dust explosions in mines is the residue of partly consumed dust. Observations after tests in the Experimental Mine have shown that the quantity of coke found after an explosion depends a great deal on the coking properties of the coal being tested. Large quantities are found after tests of Pittsburgh coal dust, and none after tests of feebly coking or noncoking coals, such as subbituminous coals. A slow explosion is favorable to the

production of coke, as the dust is exposed to heat for a longer period of time and there is less violence tending to destroy masses of it that may be formed. In appearance the coke may vary from gritty granular material to bright shiny sheets from which nearly all the volatile matter has been expelled. Quantities of individual coke bubbles are sometimes found on the floor; these are so light that they sometimes drift with a strong air current. When the flame lingers for a considerable period at one point, as where there may be accessions of air from adjacent spaces not traversed by the explosion proper, exposed coal ribs may be coked and blistered in place. This is especially apt to be the case in gassy areas. There has been sufficient variation in the position of coke deposits in different tests to make difficult the recognition of any system of formation by which the origin of an explosion in a commercial mine could be deduced from observations of the coke deposits found afterward.

In general, however, near the origin of a dust explosion the coked dust is found facing the source of explosion, when the explosion accelerates it is more apt to be on the reverse side of timbers or projections, and as an explosion dies away it is found on the facing side of projections.

Much less coke is found after violent explosions of coal-dust, even where the coal has highly coking characteristics. This may be due to more complete combustion or it may be that conditions are less favorable to its formation; also the chances that large masses will be broken up by subsequent violence are much greater.

Movement of Objects by an Explosion

In Experimental Mine tests many observations have been made of the movement of objects whose exact position was known before the explosion. The forces that are exerted in violent explosions are enormous and the destruction caused frequently resembles that following a blast of explosives. Knowledge of the mechanism by which destruction is accomplished and heavy objects carried long distances is qualitative only. Strong pressure waves can cause great destruction, but a single wave can cause only limited movement of the resulting debris. Movements of considerable magnitude are probably caused largely by columns of air or burned gases propelled through the passageways at extremely high velocities; these pick up and carry objects which sometimes weigh hundreds of pounds. The cause and power behind these movements lies in the rapid combustion of coal-dust in air - that is, in the zone in which combustion is proceeding actively. When an explosion is traveling freely along a passageway and is not dying out, the pressure in the flame produced by the intense heat is greater than that ahead or behind it. Because of this a horizontal column of air is forced ahead of the advancing combustion zone at high velocity and, similarly, a column of burned gas is propelled backward from that high-pressure zone toward the origin of the explosion. An object picked up and carried by the air blast ahead of the explosion will frequently be overtaken by the flame, which travels at a higher velocity and with rising pressure; the direction of movement of the object is reversed when the flame passes it, and, if the explosion continues strongly, it may eventually come to rest considerably nearer the origin of the explosion than it was originally. This has happened many times in the Experimental Mine. Fragments of shelving originally installed 200 to 400 feet from the face of the main entry have been found in a heap at the face of the entry completely burying the cannon used to initiate the explosion at that point. Loaded mine cars have been overturned and thrown toward the explosion origin.

At times the manometers record strong pressure waves traveling backward through the burned gases. These appear to originate in the flame and travel with the velocity of sound under the conditions of density and temperature of the after gases then and there obtaining. They have been called "retonation waves" in Experimental Mine terminology. In a mine there cannot be precise distinction between waves traveling at the velocity of sound as determined

by the temperature and pressure of the atmosphere traversed, and movement forward or backward of columns of gas caused by pressure differences. The zone of combustion sends off waves continuously. All sorts of combinations result when these waves are reflected from the rough walls and closed ends of passageways; also their velocities with reference to a fixed point change with the temperature, pressure and bodily movement of the gas through which the wave travels. It is only when a wave of unusual amplitude appears that one can trace it as it is recorded on successive manometers.

After an explosion is extinguished there is an inrush of air to fill the partial vacuum created by cooling of the hot gases. This inrush may carry light objects for considerable distances and further complicate the evidences of movements of material. It is clear, therefore, that the position of objects after an explosion cannot always be relied on to determine the point of origin of the explosion; final movement may be toward the origin or away from it, depending on how the explosion developed. It requires careful study in retracing the path or paths of a mine explosion disaster to evaluate the contradictory evidences of violence.

Relative Explosibility of Coal-Dusts

The relative explosibility of a coal-dust is measured by the proportion of incombustible material that must be present with it in a mixture to prevent propagation of flame under standard test conditions. When dust of a given coal is tested the quantity of incombustible material required to prevent propagation depends on the size and quantity of dust present, the strength of the source of ignition, the ease with which a cloud of the dust can be formed, and the configuration and physical condition of the passageways in which the tests are made. Small percentages of inflammable gas in the air current add to the explosibility of coal-dust, for the flame then travels through a mixture of coal-dust, gas, and air. It has been found possible to relate the explosibility of different coal-dusts to their volatile matter content expressed on a moisture and ash-free basis. The lower the volatile content, in general the less explosive the dust; for example, dust of the true anthracites of eastern Pennsylvania (i.e., with a volatile-total combustible ratio, of 5 to 9.5 percent) will not propagate an explosion. Semi-anthracites having a volatile ratio of 12 to 14 percent are on the border line of explosibility in the absence of gas. The dusts of coals of higher volatile content are definitely explosive.

Inflammable gas (firedamp) in the air causes an increase in the explosibility of a coal-dust cloud suspended in that air. The increase is in proportion to the amount of gas present, but its exact rate depends also on the relative explosibility of the coal-dust in the absence of gas. As a result of Experimental Mine tests the additional amount of inert material that must be kept in coal-mine dust to offset the presence of 1 or 2 percent firedamp in the air can be predicted with some accuracy without resort to special tests each time information is wanted.

The foregoing shows that determinations of explosibility of coal-dusts made in the Experimental Mine are relative and not absolute. The proper basis of correlation of the different factors that affect relative explosibility is their effect on the rate of production and dissipation of energy in the zone of combustion that supplies the driving force of the explosion. This can be done qualitatively, but experimental difficulties make a quantitative correlation well-nigh impossible, however, for practical mining application, the relative determinations are sufficiently precise to permit, for example, specifying the amount of rock dust or inert dust that is required to prevent the initiation or propagation of a coal dust explosion.

LABORATORY TESTING OF THE INFLAMMABILITY OF COAL-DUST

From its inception the Bureau has sought laboratory methods which could replace the more expensive testing carried on in the Experimental Mine. A summary of this work has been issued recently (see Bulletin 365 listed at the close of this paper). Originally attempts were made to correlate explosibility with pressure produced on inflammation of a small quantity of pure coal-dust in a closed vessel. This line was pursued for a number of years until the evidence accumulating from tests in the mine became overwhelming opposed to it. The next general line of procedure was use of laboratory "galleries" - devices designed to simulate a mine passageway on a small scale. Experimenters who endeavored to correlate explosibility with length of flame produced in a gallery were not successful. However, it is believed that a gallery could be devised in which the amount of incombustible required to suppress flame in a mixture could be determined just as it is in the Experimental Mine. At present, work is being carried on with laboratory furnaces following a method developed by the Safety in Mines Research Board of Great Britain. A weighed quantity of dust is blown downward through a vertical furnace; if it is inflammable, flame appears at the bottom of the furnace. The minimum quantity of added incombustible material required to suppress the flame is then determined. It has been found possible to arrange this test to give results comparable with those obtained in the Experimental Mine under certain fixed conditions. However, it appears unlikely that this, or any other laboratory test can completely supplant the mine; rather, they will be valuable auxiliaries to it. The present laboratory test has shown its value in giving information on the explosibility of coals and mine dusts when only small samples were available. It is also of use in investigating the explosibility of other mineral dusts like those of gilsonite and sulphur, and also of metallic dusts such as aluminum; many serious explosions of such dusts have occurred in industrial plants.

CONCLUSIONS

The foregoing paragraphs have traced the history of the Experimental Mine and the investigations conducted therein of the explosibility of coal-dust, and have given a brief account of related investigations made there. Studies of the explosibility of coal-dust have been the principal work for over 20 years, but, while a great deal of necessary and valuable information has been accumulated, many points involving the complexities of mines are still obscure and require further study. It is doubtful if the work will ever reach a point such that the conclusions drawn can be said to be scientifically final or all the practical questions are answered. Rather, the scope of the work will be determined by the needs of and changes in methods of mining, and investigations will be carried as far as the value of the data warrants. Those in charge of the work have always considered that many lines of investigation dealing primarily with increased safety for miners should be carried on as need of them arises, and a number of such researches have been cited in the foregoing pages. They look upon the Experimental Mine as a large-scale laboratory in which problems in mine safety can be investigated under limits prescribed only by the facilities available and the nature of the information desired.

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INFORMATION CIRCULAR
UNITED STATES BUREAU OF MINES

LIMESTONE

Part II. - Dimension Stone¹

By Oliver Bowles²

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2 Supervising engineer, building materials section, U. S. Bureau of Mines.

FOREWORD

Bureau of Mines Information Circular 6723, entitled "Limestone: Part I. - General Information", discusses the origin and nature of limestone; shows production figures by States and uses; and describes in some detail the quarrying, preparation, uses, and marketing of crushed, broken, and pulverized limestone. Dimension stone, the other major branch of the limestone industry, is considered in this paper. Travertine, which is classed with limestone, is discussed in an Information Circular, in press, entitled "Onyx Marble and Travertine".

DEFINITION

Dimension stone is a general term applied to blocks of stone which with the exception of rubble and boulders are cut to definite shapes and often to specified sizes. The principal limestone products of this class are cut, carved, and roughhewn or scabbled blocks of building stone. Rubble, rough and irregular wall stone, is included with dimension stone because it is employed in masonry walls. Dimension stone is contrasted with crushed, broken, and pulverized products, which consist of irregular fragments sized chiefly by mechanical screening or air separation.

PRODUCTION

Table 1 shows production figures of dimension limestone by kinds over a series of years.

Table 1. - Dimension limestone sold in the United States, 1922-31

Year	Building stone ¹		Curbing, flagging, and paving		Rubble		Total value
	Cubic feet	Value	Cubic feet	Value	Short tons	Value	
1922	12,617,380	\$12,418,873	(2)	(2)	286,340	\$470,264	\$12,889,137
1923	14,639,950	16,221,095	75,070	\$46,830	327,180	484,771	16,752,696
1924	14,456,320	15,805,680	69,010	78,264	392,180	623,844	16,507,788
1925	15,983,800	16,092,079	129,730	98,587	324,630	513,387	16,704,053
1926	18,537,950	20,391,597	167,780	135,832	254,240	476,545	21,004,024
1927	17,340,690	18,820,045	223,370	134,360	226,280	400,790	19,355,195
1928	17,641,370	20,193,963	322,560	205,724	365,920	705,723	21,105,410
1929	17,864,700	20,649,257	471,880	158,266	352,480	693,678	21,501,201
1930	15,682,720	18,535,293	346,040	137,801	756,470	623,100	19,296,194
1931	11,706,840	10,858,697	166,260	85,175	229,510	296,426	11,240,298

1 Small quantities of monumental stone are included.

2 Included with building stone.

It will be observed that the value of building stone amounts to 95 percent or more of the total value. Rubble, while used for building purposes, is classed separately. Material classed as building stone in this table may be subdivided according to its character and the extent to which it is manufactured into finished products. Table 2 shows the quantity and value of the various classes of building stone sold during recent years.

Table 2. - Building limestone sold in the United States by kinds,
1922-31

Year	Rough construction		Rough architectural		Finished (cut and sawed)		Total value
	Cubic feet	Value	Cubic feet	Value	Cubic feet	Value	
1922	210,690	\$271,343	4,352,030	\$2,577,416	5,843,550	\$9,570,114	\$12,418,873
1923	210,310	238,805	5,031,840	3,184,932	7,192,810	12,797,358	16,221,095
1924	239,680	304,911	5,012,520	3,231,613	6,751,930	12,269,156	15,805,680
1925	270,970	335,842	5,728,830	3,661,276	7,068,070	12,094,961	16,092,079
1926	288,700	380,131	6,161,390	3,837,081	9,015,090	16,174,385	20,391,597
1927	206,480	292,792	5,953,780	3,832,414	8,811,110	14,694,839	18,820,045
1928	211,780	279,434	5,255,690	3,268,140	10,655,180	16,646,389	20,193,963
1929	178,490	230,564	6,192,550	3,779,639	9,430,100	16,589,054	20,649,257
1930	131,410	194,550	6,134,250	3,278,346	7,936,770	15,062,397	18,535,293
1931	220,970	317,852	3,594,270	1,574,720	5,378,810	8,966,125	10,858,697

Rough construction stone, the first of the three groups of building stone in this table, is sold mostly by the ton for rustic building of many kinds. It overlaps to some extent the material classed as rubble in table 1.

Rough architectural limestone is sold principally to independent fabricating mills that buy rough stock from quarry companies. Material classed as finished stone is that fabricated in mills operated by quarry companies. Thus, the total value in the last column of table 2 is made up partly of rough-block and partly of finished-product values. This is unavoidable because the Bureau's statistics are based on sales made by the original producer. According to figures compiled by the Bureau of the Census, the total finished-product value of building limestone sold by fabricating plants in the United States in 1929 was \$51,301,968 and in 1931, \$24,142,920. These values are more than twice those shown in table 2. The difference is due to the added value of finished stone manufactured in independent mills.

REQUISITE QUALITIES

While innumerable limestone deposits occur throughout the country, only a few consist of rock that will satisfy the exacting requirements of dimension stone. Deposits with irregular or closely spaced joints are not suitable because large blocks free from cracks or lines of weakness are generally demanded. Limestones display great variation in physical properties, such as texture, porosity, hardness, strength, and color, and upon these depend their adaptability and value as dimension stone. The chemical composition also varies in different types of stone, but while this may have some bearing on the quality of the material, it is of little consequence in this branch of the industry.

Dimension stone must have adequate tensile and compressive strength. Sound structural limestone suitable in other respects is almost invariably strong enough for any use; in fact, the strength of ordinary stone far exceeds the requirements for safety. However, it is generally conceded that rock disintegrates and weakens more readily when under severe stress, and therefore a factor of safety of 20 is demanded; that is, stone must be able to resist a crushing stress 20 times as great as that to which it will be subjected when placed in a wall. Tests of transverse strength - that required to sustain a load applied at the middle of a bar of stone supported at the ends - are more important than crushing-strength tests for dimension stone.

The suitability of a limestone depends to quite an extent on the degree of cementation of the grains. If they are loosely coherent the limestone is "earthy" or "friable". Material in which they adhere closely and strongly is "compact". Most limestones may be regarded as sufficiently durable for all ordinary uses, but the heavier or compact types are most desirable. Weight per cubic foot is a fair index of durability.

Hardness has a direct bearing on workability. While all limestones are soft enough to be scratched easily with a knife the presence of flint or other siliceous minerals may make quarrying or milling more difficult. For stone that is to be subjected to abrasion, such as paving, curbing, floor tile, or stair treads, hardness is a requisite quality, whereas it is not necessary in material used for walls or decoration.

For most uses uniformity of texture and color is preferable. Coarse texture was once regarded as a serious imperfection, but tests have shown that coarse-grained stone compares favorably in durability and strength with fine-textured material, and modern demands for variety have led to its wider use.

Limestones are of many colors, ranging from white to black. Colors other than white are produced by impurities in the stone. Certain constituent minerals, such as pyrite, marcasite, or siderite, may oxidize by weathering and cause surface stains which are serious blemishes. The colors of some limestones change with age, and while this is not always objectionable permanence of color is usually preferred. Choice of color depends on environment, individual taste, or prevailing fashion. Brown, buff, gray, or white limestones



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are usually chosen for building purposes. White and other light colors are not desirable for smoky cities. Variegated stone, sawed and rock-faced, is now popular for residences.

ECONOMIC CONDITIONS

A deposit of limestone may meet all the general requirements as to quality, but development may not be justified from a business standpoint. Availability of markets is an important consideration. Stone being a heavy product, the freight charge is a material part of the delivered price, and consequently a location within easy reach of large centers of building construction is to be desired. Transportation facilities, by truck, rail, or water, should be favorable. The best rates are obtained where two or more competing lines are easily accessible. Cost of production has definite bearing on market range, for a low-cost product can absorb a relatively high freight charge and thus compete successfully in distant markets with high-cost local stone. Quarry conditions should be considered carefully before development is undertaken. A thick overburden of soil or rock or a heavy flow of water requiring excessive pumping may constitute large items in the cost of production. Availability of labor, living conditions, and prevalent wage levels are other matters that deserve attention.

USES

Dimension limestone is used principally for building purposes. Its very limited employment for monuments, curbing, and flagging may almost be disregarded. Building limestone may be grouped into four classes of products - cut stone, ashlar, rough building stone, and rubble. The largest quantity consumed is in the form of cut or rough-hewn blocks.

Cut Stone

Cut stone includes all forms of finished blocks, columns, sills, moldings, balusters, or special designs. It is the most expensive type because usually the blocks are accurately shaped and sized in accordance with detailed drawings. Cut blocks may be used for entire buildings or only for certain parts, as window sills, trim, cornice, or base course. Massive blocks are used for bridges, dams, sea walls, and similar structures. Columns and balusters are used on both the inside and outside of buildings. For decorative purposes blocks may be elaborately carved.

Ashlar

Ashlar is a term applied to small rectangular blocks having sawed, planed, or rock-face surfaces. Several types are used. "Even-course" ashlar consists of blocks usually 3 or 4 inches thick of uniform height for each course, though succeeding courses may be thicker or thinner. They may be of uniform or random lengths. Two or more unit thicknesses may be used, several of the smaller sizes giving the same height as one of the larger blocks. "Random" ashlar consists of blocks that may be fitted together in a wall having irregular and unequally spaced joints.

Ashlar is particularly adaptable for residences and is a much cheaper product than cut stone. Its production is also economically advantageous to producers, for small material that would otherwise be wasted may thus be utilized.

Rough Building Stone

Rough building stone consists of rock-faced masses of various shapes and sizes. It is used extensively for chimneys, basements, or entire walls of residences and to some extent for public buildings, bridges, fences, and the more ornamental types of retaining walls.

Rubble

The term "rubble" is generally applied to irregular fragments having at least one good face. It is the crudest form of building stone. Many years ago rubble constituted a distinct type of stone that was used principally for foundations and other rough walls, but concrete has gradually replaced stone for such types of construction. On the other hand, fashion in architecture was changing meanwhile from smooth and formal walls to rougher and more variegated styles, with the result that rubble has attained more dignified uses. From its lowly cellar-wall position it has been elevated to share with cut stone a more prominent place in many types of construction. Therefore, while rough construction stone and rubble were at one time fairly distinct types they are gradually merging, and it becomes increasingly difficult to differentiate them.

CENTERS OF PRODUCTION

Although limestone occurs in every State, deposits suitable for dimension stone or conditions favorable for their development exist in only about a half of the States.

Alabama is an important producer of oolitic limestone. Recent developments are chiefly near Rockwood where a large mill provided with the most modern equipment is operated. Shell limestones are quarried for building stone in Volusia and Pinellas Counties, Fla.

The largest quarry center for building stone in the United States is the oolitic limestone district near Bedford, Lawrence County, and Bloomington, Monroe County, Ind. In this district more than 20 large companies operate between 30 and 40 quarries and mills, and about 12 others operate finishing mills only. In 1930, exclusive of rubble and a small quantity of stone sold for rough construction, figures for which are not available, Indiana supplied 12,702,980 cubic feet valued at \$16,186,172, or more than 81 percent of the total quantity and 87 percent of the total value for the United States. The total value as here shown does not include the value of finished products from independent mills. The 1930 figures are given because they are more typical of average conditions than those for 1931 or 1932. The Salem formation,

which provides the stone, is a massive bed ranging from 25 to nearly 100 feet in thickness. It extends from the Ohio River northward through Salem, Bedford, and Bloomington to a point north of Greencastle, a distance of about 125 miles. Buff and gray limestones predominate. The stone is easily worked when first quarried but becomes harder when seasoned.

Kentucky furnishes an oolitic limestone similar to the Indiana stone. It is quarried in Warren County and shipped to mills in Bowling Green for manufacture. Quite a number of quarries have been worked at various times, but only two have been operated during recent years.

A dolomitic limestone quarried principally at Mankato and Winona, Minn., is adaptable for heavy masonry, bridge construction, and building stone. An ornamental variety quarried at or near Kasota in the same State is widely used for interior decoration.

A pale buff to cream oolitic limestone is quarried for building purposes at Cedar Park, Williamson County, Tex. Other producing districts in Texas are Lueders, Jones County, and near Del Rio, Kinney County.

Limestone is also quarried for building purposes in a small way in Colorado, Illinois, Kansas, Maryland, New York, Ohio, Pennsylvania, Utah, and Wisconsin.

PROSPECTING

Development work should not be started on a deposit without reasonable assurance that an abundant supply of limestone suitable for dimension stone is available. Generally it is deemed unwise to expend the large sum of money necessary to establish quarries and finishing plants unless a reserve for at least 20 years' operation is assured. Core drilling is usually desirable for determining the extent and quality of a deposit. As quality may vary in different beds, each one should be tested. The essential features, such as color, grain, and extent of beds, can be determined from the cores. No definite rules can be given for the position or arrangement of prospect holes. In Indiana they are spaced 100 to 300 feet apart or closer in some instances. A log should be kept of the character of limestone found in each hole, and the cores should be marked and preserved for future reference. Depth of overburden may be determined by drilling or trenching.

QUARRY METHODS

Quarrying and milling methods discussed herein are those employed in the limestone district of Indiana, but they are essentially the same in other districts.

Plan

Most deposits of limestone used as dimension stone are approximately flat lying and of limited thickness. Therefore, the stone available in any one opening may be removed within a short time and a new ledge uncovered. Ledges are usually 120 to 140 feet wide - a width that can be served with a derrick boom. In large quarries a series of derricks is set in line, and a long ledge is worked down in a succession of floors until all the good stone is removed. The line of derricks is then moved back 130 to 140 feet and another ledge begun; thus a whole area may be worked out in succeeding strips. Waste material from later operations is thrown into the openings previously made.

Stripping

In some places the overburden is stripped by power shovels. The hydraulic method is employed, however, when the surface contour is favorable for washing the soil into abandoned pits or other low-lying areas. This method is especially advantageous for washing clay from mud seams.

A second phase of stripping involves the removal of overlying noncommercial rock, which in some quarries may extend to a depth of 60 feet. The method of removal is governed chiefly by the nature of the contact of this rock with the underlying stone. Where it is separated by a layer of clay, shale, or a loose bed that serves as a cushion it may be drilled and blasted with light charges of low-grade black blasting powder with little danger of shattering the good material. However, where the rock overburden is continuous with the stone of good quality this process cannot be used, for powder cracks will extend into and destroy the good stone beneath. In such cases it may be necessary to channel the waste and remove it in block form - a stripping process almost as expensive as the removal of commercial stone. A unique process has recently been developed whereby a horizontal wire-saw cut is made at the base of the top rock. This artificial seam protects the stone beneath and permits removal of the top rock by blasting.

Channeling

Channeling machines make the primary cuts for block separation. The machine operates with a chopping action similar to that of a reciprocating drill, and cuts a channel or narrow trench on one or both sides as it travels back and forth on a 7-foot 2-inch-gage track. The edge of the cutting tool is 1-7/8 to 2-1/8 inches wide and makes a channel about 2 or 2-1/4 inches wide. Cuts are 8 feet 4 inches to 8 feet 5-1/4 inches apart and may be 8 to 12 feet deep and 50 to 100 feet long.

The cost of channeling ranges from 8 to 12 cents a square foot; it is, in fact, the largest single item of quarry cost and may amount to more than half the total cost.

Wire Sawing

The high cost of channeling has led some operators to attempt more economical methods of cutting. At least two companies in Indiana are now using wire saws with fair success. One company made quite exhaustive tests in 1931. A cutting rate of 87 square feet an hour was attained, and the average cost was 13.9 cents a square foot. Details have been published by Newsom³, who directed the work.

Key-block Removal

In opening up a new floor where no free face is present the removal of the first or key block is difficult. Sometimes it has to be broken up and removed as waste.

After an 8- by 8-foot block has been channeled on four sides the tracks are shifted and then two 2-foot blocks are channeled. The latter, known as "pulling" blocks, are usually comparatively easy to break loose by wedging in the channel cut. When a block is free, dogs or hooks are used to hoist it out. When one block is out of the way floor space is provided for removing succeeding blocks, which are wedged free at the floor and removed one by one.

Another method of starting a new floor is to remove a mass 2 feet wide along the wall and make a long cut 20 feet from the 2-foot space and cross-cuts 4 feet apart. After the pulling blocks are removed the 4- by 20-foot masses are turned down in the usual fashion.

Bed Lifting

Masses of rock that have been channeled are separated at the floor line by drilling and wedging. For this purpose a series of holes 8 to 12 inches deep and 1 foot to 18 inches apart is sunk with air-driven hammer drills. They are not made at right angles to the wall but are inclined at such an angle that wedges placed in them may be sledged conveniently. Wedges are placed in the holes and driven in succession until a floor break is made.

Turning Down Blocks

After a block is wedged free at the floor it is turned down in a horizontal position before further subdivisions are made. Two notches or dog holes are made in the back channel cut to accommodate massive hooks, which are connected by means of sheave and tackle to another pair of hooks secured firmly on the quarry floor some distance in front of the block. A heavy strain exerted on the cable by a derrick hoist gradually pulls the block over.

³ Newsom, J. B., Results of Wire-Saw Tests: Trans. Am. Inst. Min. and Met. Eng., 1932, pp. 117-121.

Large wedges may be sledged in the back channel cut to assist the process. Piles of loose rock, "pillows", are placed to break the fall so that the block comes to rest with little impact and without danger of breaking. Masses thus turned down are usually about 60 feet long, 12 feet high, and 4 feet wide.

Block Subdivision

The next step is to divide the mass into sizes for commercial use. Blocks are laid out with a carpenter's square and straightedge. Where two or more grades of rock are present longitudinal breaks are made between them. Holes are first drilled in line 6 to 8 inches deep and 12 to 18 inches apart. Fractures are then made by the "plug-and-feather" method. The "feathers" consist of strips of iron which are flat on one side for contact with the wedge and curved on the other to fit the wall of the drill hole. Two are placed in a drill hole, and a "plug" - a steel wedge about 6 inches long - is driven between them. They are sledged lightly in succession until a fracture appears. Blocks are commonly 10 by 4 by 3 feet and 10 by 4 by 4 feet.

Hoisting

Steel or wooden derricks of about 30 to 50 tons capacity are used for hoisting blocks from quarries. Derricks now in use have masts 80 to 110 feet high and booms 70 to 100 feet long. They are the swinging-boom type and are supported by 10 to 15 guy cables secured to deadeyes in the rock or attached to buried timbers. The end block is raised about 3 feet at the outer end and lowered again to the floor. This operation crowds it outward and makes a space of a foot or more for attachment of dogs. Heavy hooks (dogs) are inserted into the "dog holes" cut on opposite sides of a block, and a chain passed through the eyes of the hooks draws them firmly against the block, thus holding it securely for lifting. Blocks are removed in succession and placed on cars or piled for later disposal. Positions for attachment of dogs should be carefully chosen so that blocks may be exactly balanced. Each block is numbered or lettered to indicate its classification and for office records.

Storage and Transportation

Large storage capacity is essential, for enough stone must be accumulated during summer months to supply the demands of the four winter months when quarries are idle. Outdoor storage or "stacking yards" may be maintained at quarries or mills or at both places. A common method is to pile blocks within reach of derrick booms. They are usually piled high in a limited area, and it is sometimes difficult to sort them. Some operators prefer overhead-traveling-crane storage because the blocks may be handled more quickly. Standard railway cars and locomotives are almost invariably used for haulage.

Scabbling

Quarry companies that operate mills use rough, untrimmed quarry blocks. If no local mills are operated, the blocks may be shipped long distances, and they must be trimmed carefully to avoid freight charges on waste. The trimming process is called "scabbling". Several methods are employed.

Scabbling picks similar to the ordinary double-pointed miner's pick are used to remove all irregularities. They are used also to remove corner masses from blocks to be turned into columns. Many operators prefer scabbling saws of various types because they leave a smooth, even surface and in a single operation remove large projections which by the pick method are taken off piecemeal. Diamond-tooth drag saws are used singly or in parallel pairs adjustable for width. Diamond-tooth circular saws cut rapidly and if mounted in pairs adjustable for spacing may scabble both sides of a block at one time. However, the depth of cut is a limitation; a 60-inch saw can cut only 26 or 27 inches and a 72-inch saw, 32 or 33 inches. This difficulty is overcome by making one pair of cuts to the maximum depth the saws will reach and then turning the block over and cutting from the reverse side. If the cuts fail to meet, the intervening rock may easily be broken.

Scabbling planers are effective substitutes for saws. They consist of two sets of massive vertical blades which scrape off irregularities as a block passes between them on a bed. By screw-feed adjustment the blades are set closer after each cut until a smooth surface is obtained. On blocks 6 feet high each cut removes one fourth inch of rock and on blocks 4 feet high one half inch. About three blocks may be scabbled in an hour by this method.

A wire saw consisting of a 3/16- or 1/4-inch 3-strand cable running as an endless belt driven by an electric motor is also used for scabbling. Several blocks may be lined up and cut at the same time. The equipment may be operated by one man, and an average cutting rate is 20 to 25 square feet an hour.

MILLING METHODS

Some limestone companies quarry only, others also maintain mills where the stone from their quarries is sawed into slabs or manufactured into finished forms, while still others operate only mills and buy their rough material for fabrication.

Drafting and Pattern Making

Before any cut-stone job can be started, accurate detailed drawings must be made of every piece of stone to be used that differs from another in size or shape. The cost of the drafting work required for a large structure may be one half to two thirds as much as the entire quarry expense of supplying rough blocks of stone. Architects' drawings usually are insufficient, for

the stone must be fitted accurately to the steel framework, and detailed data on the size and position of each steel member are necessary before stone workers' shop drawings can be made. Some sizes may be duplicated many times in a building; others may be used only once or twice.

Patterns for molded and carved work are made of zinc or other soft metal, or sometimes paper patterns or stencils are used, the design being transferred to the stone surface by dusting burnt umber through perforations. For the most intricate carved work plaster models are used.

Ticket System

After shop drawings are made draftsmen prepare for every block of stone a ticket on which is a drawing of the block with exact dimensions indicated. The ticket is numbered, and if a pattern is to be used its number is also shown. The man in charge of gang sawing first gets the ticket and cuts the block or slab. As this passes to the planer, jointer, and all subsequent machines the ticket goes with it, and each workman consults it before any work is begun. By this means workmanship is constantly verified, and very few mistakes occur. Great care and skill are required, for one small error may ruin a block.

Block Handling

Mills are of two types. One is wide and equipped with two pairs of crane tracks. A heavy crane for placing quarry blocks on saw beds travels on one track; and on the other, a lighter and more rapidly moving crane for conveying smaller blocks as they pass from one operation to another. The crane almost universally used is the overhead traveling type with at least a 70-foot span.

The other type of mill is long and narrow with one pair of tracks on which several cranes operate. There may be a 25-ton-, 15-ton-, and a $7\frac{1}{2}$ -ton-capacity crane working at the same time. Most of them are of the 2-motor type, one motor with two friction clutches serving for both lateral motion and hoist.

Railway tracks for transporting quarry blocks to mills and shipping finished products enter across the end, down one side, or across the middle of the building. Rough blocks and single unfinished slabs are handled with grabhooks; finished and semifinished blocks or piles of slabs, with cable slings or with slings of rubber belting so as to avoid damage to corners and edges.

Operators travel back and forth in cabs attached to the crane. Two men called "hookers" attach and release hoisted blocks and signal the craneman.

Sawing

The first step in manufacture is to saw rough blocks to requires dimensions. Gang saws of various sizes are used almost universally for this purpose. They consist of a series of soft steel blades set in parallel positions in a frame which has a backward-and-forward swinging motion. As the frame moves, actuated by a crank and connecting rod (pitman), the cutting blades lift toward the end of each stroke. This permits the abrasive, usually sand, to wash under them, and as they start back on the return stroke they bear down on the sand and abrade the rock rapidly. An adjustable automatic gear feeds the gangs downward at any desired rate. Steel shot is used as abrasive when a deeply scored, "ripple-mark" surface is desired. The average cutting rate is about 6 inches an hour. The blades may be spaced for thin slabs or thick blocks.

Another type, a straight steel blade with diamond teeth on the lower edge, is used as a drag-saw for making single cuts. A drag tooth is mounted with six diamonds of about 3/4-carat size placed in alternate positions on opposite sides of the cutting face. This saw will cut at a rate of 30 to 40 square feet an hour.

For making subsequent cuts circular diamond saws are almost invariably used. The blades are steel one fourth inch thick with a series of square notches around the rim. Steel teeth mounted with diamonds are set in the notches and held in place with copper rivets.

A heavy service type known as a "ripsaw" has a stationary mounting, and a bed actuated with a worm gear carries the block of stone beneath it. Teeth for ripsaws are supplied with two 1/2- to 5/8-carat diamonds. A jointing saw designed for smaller cuts and similarly mounted is provided with teeth, each having 6 to 10 small diamonds. A stream of water cools the cutting edge of a diamond saw and carries away the cuttings. The sawing rate is 3 to 16 inches a minute, depending on the depth of the cut. The most exacting workmanship is exercised in the manufacture of diamond circular saws to insure accurate balance, uniform cutting, and true running. Each saw is designed for a standard speed and should be run at no other. Speeds range from 11,000 to 13,000 surface feet a minute. With care a saw will perform constant service for 6 months to a year without being conditioned. Although the price of a diamond saw is high it cuts rapidly, and maintenance cost is low.

Silicon carbide (Carborundum) circular saws give excellent service for the smaller cuts because they leave smooth surfaces and are less liable than are diamond saws to chip the corners of blocks.

Planing

Planers are used for cutting stone blocks and slabs to smooth surfaces and desired thicknesses and also for cutting moldings and shaping curved designs. The frame that holds the cutting tool has lateral and vertical motion

actuated by power-driven worm gear and may cut the tops and sides of a block simultaneously. As a block of stone is carried beneath a planer on a traveling bed called a "platen" at a rate of 30 to 45 feet a minute, the cutter scrapes a thin layer from the surface. Cutting tools are shaped to fit molding designs; that is, the tool is the reverse of the pattern.

When curved designs are to be cut the stone is placed on a second bed or platen resting on the regular planer bed and capable of rotating through an arc of a circle. This second platen is guided by an arm, pivoted to one side of the planer, whose length governs the curvature of the arc.

A Carborundum planer consists of two saws with a drum of smaller diameter between them. The saws trim the sides of slabs while the drum smooths the upper surfaces. The planer bed travels at a rate of only 20 to 30 inches a minute but finishes the job in one cut, thereby accomplishing much more in a given time than an ordinary planer.

The accomplishment of a planer is equivalent to that of seven stone-cutters using hammers, chisels, and modern pneumatic tools.

Turning and Fluting

Large columns are first scabbled to cylindrical shape and then mounted in lathes where they rotate against a tool traveling slowly back and forth over the full length of the stone. The tool is actuated by machine-driven worm gear and may be adjusted for any diameter by a screw feed. A smooth surface is obtained by this process, but final rubbing is usually done by hand.

The fluting process is also carried out on a lathe. The flutes are drawn with pencil or crayon on the surface of the column, and the fluting tool attached to the tool post of the lathe travels back and forth while the column remains stationary. This process is continued until the line bounding the flutes is reached. If a column is tapered the flutes may be cut shallower on the smaller parts of the column, and this automatically makes them narrower. When a flute is completed the column is rotated with a hand bar, and the process is repeated in the new position. The ends are finished with pneumatic tools, and the column is rubbed by hand.

For making balusters, particularly if many of one kind are to be fabricated, a Carborundum wheel cut as a negative of the pattern is generally used.

Milling

"Milling" is a term applied in a general way to all mill processes, such as sawing, planing, cutting, or carving, but it specifically refers to the operations of a milling machine by which irregular patterns are cut in stone. Milling is particularly advantageous in preparing for the carvers blocks in

which deep recesses are to be cut, for it removes the bulk of the stone much more rapidly than do hand processes. The machine consists essentially of a rotating head which carries cutting tools of various sizes and shapes. It has right-and-left as well as vertical worm-gear motions, and a movable platen provides backward-and-forward motions.

Cutting and Carving

Cutting usually is defined as straight-line work and carving as curved work. Carving requires much skill and is generally done by experienced workmen. The work is now done for the most part with pneumatic tools - heavy ones for removing the larger fragments and smaller ones for completing designs, but a chisel and mallet are still necessary for certain operations. Patterns are used for the designs and models for the complicated figures.

Oolitic limestone may be carved more easily than most other building stones and therefore can be finished at a cost that permits of extensive use.

Finishing

Various types of surface finish are achieved by the use of different kinds of tools. A tooled surface, which is one covered with fine grooves in parallel lines, is made with a pneumatic or planer tool having fine teeth. A bush-hammered surface is rough and pitted, the hammer used having a face covered with small projections. A hand-picked surface is covered with indentations made with a sharp-pointed tool. A small-fluted surface is covered with small parallel corrugations. A four-cut surface is made with a planer tool having four corrugations to the inch. A rubbed surface is smoothed by hand-rubbing with sand and water or some other abrasive. A shot-sawed or ripple surface is deeply scored or grooved by using steel shot as abrasive for the gang saws. Chat-sawed stone is smoother than the shot-sawed stone.

COST OF QUARRYING AND MANUFACTURE

Quarrying and milling costs vary, as they depend on diverse conditions, such as depth of overburden, degree of hardness of the rock, type of equipment used, working efficiency, skill of workers, and size of operation. Approximate costs only may be given.

Quarry costs range from 20 to 30 cents per cubic foot of stone. Mill costs are extremely variable because only a little work is expended on some blocks and a great deal on others. Sawing slabs is expensive, costing 35 to 45 cents per cubic foot. Sawing in a second direction costs 12 to 15 cents more. Planing, milling, and cutting costs must be added for most products. Carving is also expensive. For jobs requiring a moderate amount of work, the total cost ranges from \$1.50 to \$2.50 per cubic foot. Columns or curved designs may amount to much more. Gothic carving may cost as much as \$7.50 per square foot of surface carved.

MARKETING

Dimension stone is sold by the cubic foot, and much of it commands a price sufficiently high to give it a Nation-wide market. From two thirds to three fourths of all building limestone is sold as rough blocks or sawed slabs to mills in consuming centers, where it is fabricated chiefly for small or moderate-size buildings. The balance of the production is manufactured for specific projects, usually of large size, in mills operated in conjunction with quarries or in independent mills. These mills, supplied only with shop drawings, can furnish for a structure many miles away thousands of blocks of limestone in exact dimensions, each fitted accurately for its particular position in the wall. The smaller limestone quarries for the most part sell their stone to local builders and contractors.

Sample

UNITED STATES BUREAU OF MINES
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Jan 26*

REVIEW OF FINE GRINDING IN ORE CONCENTRATORS

This paper presents the results of work done under a cooperative agreement between the U. S. Bureau of Mines and the Missouri School of Mines and Metallurgy, Rolla, Mo.



BY

ALEXANDER M. GOW, MORRIS GUGGENHEIM, AND WILL H. COGHILL

INFORMATION CIRCULAR

UNITED STATES BUREAU OF MINES

REVIEW OF FINE GRINDING IN ORE CONCENTRATORS¹

By Alexander M. Gow,² Morris Guggenheim,³ and Will H. Coghill⁴

INTRODUCTION

This paper summarizes a survey of fine-grinding practice in ore-concentrating plants throughout the United States, Alaska, Canada, Mexico, and Cuba as reported in the Bureau of Mines Information Circulars on milling methods. The survey covers 49 concentrators, which have a combined capacity of over 160,000 tons per day. In these plants 470 ball, rod, and pebble mills prepare material for flotation, gravity concentration, amalgamation, cyanidation, and air separation. The ores are those of gold, silver, copper, lead, zinc, molybdenum, feldspar, and pyrite. The metallurgy is that of the period 1928-32.

Of these 49 concentrators, 36 are in the United States. They can treat 134,400 tons per day, and they have 326 fine-grinding mills. In the United States there are about 100 active or temporarily idle concentrators, exclusive of the many plants in the Tri-State zinc district, the washeries of the Lake Superior iron ranges and the Florida phosphate area, and small gold and silver concentrators. These 100 plants have an estimated capacity of 290,000 tons per day and employ about 650 ball, rod, and pebble mills. Consequently, since this paper covers about half of the production of this country it may be considered as particularly representative of fine grinding in ore mills in the United States.

It is the aim of this paper to facilitate comparison between the mills of different localities. For this purpose the data have been arranged in tables to which the text should be regarded as supplementary.

Table 1 lists the concentrators and presents general information regarding the fine grinding sections; table 2 shows products and costs; and in table 3 the grinding mills are listed by size. Appended is a bibliography of information circulars listed by key numbers and with the reference names underscored.

This paper represents the result of cooperative work between the Mississippi Valley Experiment Station of the U. S. Bureau of Mines and the Missouri School of Mines and Metallurgy, Rolla, Mo.

GENERAL INFORMATION

Considerable variation in fine-grinding flow sheets and in the many methods of ball-mill operation is shown in this review. Obviously, there is no standardization even in concentrators in the same district. Contributing to this condition are many factors, including the type of coarse and intermediate crushing machinery, the available fine-grinding equipment, the original design of the concentrators, the tonnage to be treated, the degree of

¹ The Bureau of Mines will welcome reprinting of this paper, provided the following footnote acknowledgment is used:

"Reprinted from U.S. Bureau of Mines Information Circular 6757."

² Assistant metallurgist, U.S. Bureau of Mines.

³ Junior metallurgist, U.S. Bureau of Mines.

⁴ Supervising engineer, ore-dressing section, Metallurgical Division, U.S. Bureau of Mines.

fineness necessary for liberation, the grade of the finished product, the general local practice, and the resistance to grinding of the ore.

In a few circulars [United Verde (5), Britannia (19), and Pecos (34)] the relative hardness and toughness of the ore have been indicated, but the industry still is in need of a universally approved method of determining a "grindability factor" so that ores may be compared as to their relative amenability to comminution.

Table 1.- Operating data of fine grinding mills

(Section A)

COPPER ORES

No.	Property	Plant rated tons per day	Type of flow sheet	Grinding mills			Circuit	Speed	
				No.	Kind	Size		R.p.m.	Per- cent crit- ical
1	2	3	4	5	6	7	8	9	10
1	Hayden	10,500	F	14	Ball	7 feet by 10 feet	Closed	16	55
2	Cananea	2,400	Ar + D and Dr + F	3	Rod	6 feet by 12 feet	Pri. op./cl.	16	48
				4	Ball	8 feet by 28 inches	Sec. closed	16	58
				2	do.	do.	do.	18	64
3	Harmony	200	D	1	Ball	7 feet by 36 inches	Closed	26	86
4	Magma	750	D + conc. + E	3	Ball	6 feet by 4½ feet	Pri. closed	25	78
				3	do.	5 feet by 10 feet	Sec. closed	26	72
5	United Verde	1,500	D + E	1	Ball	8 feet by 12 feet	Pri. closed	18	67
				2	do.	do.	Sec. closed	16	59
6	C. & H. recl.	3,000	Dp	64	Pebble	8 feet by 18 inches	Closed	26	93
7	Nacozari	2,700	Cr + Dr	6	Rod	6 feet by 12 feet	Pri. open	18	55
				6	do.	do.	Sec. closed	18	55
8	C. & H. congl.	2,900	Ap	12	Pebble	8 feet by 72 inches	Open	24	85
9	Hurley	13,000	Cr + D	9	Rod	7 feet by 10 feet	Pri. open	20	66
				18	Ball	7 feet by 10 feet	Sec. closed	20	66
10	Copper Queen	4,000	Cr + D	8	Rod	6 feet by 12 feet	Pri. open	18	55
				8	Ball	6 feet by 12 feet	Sec. closed	18	55
11	Morenci	5,000	F + conc. + E	3	Ball	7 feet by 10 feet	Pri. closed	18	60
				3	do.	8½ feet by feet	do.	14	53
				5	do.	6½ feet by feet	Sec. closed	19	62

Table 1.- Operating data of fine grinding mills - Continued

(Section A - Continued)

COPPER ORES - Continued

No.	Property	Plant rated tons per day	Type of flow sheet	Grinding mills			Circuit	Speed	
				No.	Kind	Size		R.p.m.	Per- cent crit- ical
1	2	3	4	5	6	7	8	9	10
12	Old Dominion	1,500	B + F, and Dr	4	Ball	6 feet by 50 inches	Pri. open	-	-
				1	do.	do.	Sec. closed	-	-
				1	do.	8 feet by 36 inches	do.	-	-
				1	rod	6 feet by 12 feet	Closed	18	56
13	Arthur and Magma	54,000	B + E, and E + F	13	Ball	7 feet by 10 feet	Pri. open	21	68
				39	do.	do.	Sec. closed	20	66
				18	do.	do.	Pri. closed	21	68
				36	do.	do.	Sec. closed	20	66
14	Verde Central	400	D	2	Ball	8 feet by 36 inches	Closed	15	53
15	Matahambre	1,300	D + E	4	Ball	6 feet by 4½ feet	Pri. closed	25	78
				4	do.	6 feet by 6 feet	Sec. closed	25	76
				1	do.	5 feet by 5 feet	do.	29	80
16	Engels	1,500	C + D	3	Ball	8 feet by 36 inches	Pri. open	20	72
				1	do.	8 feet by 6 feet	Pri. part. cl.	21	78
				2	do.	6 feet by 12 feet	Sec. closed	24	73
				1	do.	6 feet by 10 feet	do.	23	70
				1	do.	8 feet by 36 inches	do.	17	60
17	Walker	1,700	D	3	Ball	7 feet by 5 feet	Closed	24	83
				1	do.	7 feet by 7 feet	do.	24	83
18	Miami	17,000	F' + D + E and E, and F	12	Ball	8 feet by 36 inches	Pri. part. cl.	20	72
				12	do.	do.	Sec. closed	20	72
				6	do.	10½ feet by 8 feet	Tert. closed	13	52
				2	do.	6 feet by 12 feet	Retr. closed	23	70
				1	do.	6 feet by 22 inches	Mid. closed	25	76

Table 1.- Operating data of fine grinding mills - Continued

(Section A - Continued)

COPPER ORES - Continued

No.	Property	Plant rated tons per day	Type of flow sheet	Grinding mills			Circuit	Speed	
				No.	Kind	Size		R.p.m.	Per- cent crit- ical
1	2	3	4	5	6	7	8	9	10
19	Britannia	6,500	Modified A + E, and C + E	6	Ball	6 feet by 11 feet	Pri. open	24	73
				18	do.	5½ feet by 9 feet	Sec. closed	20	58
				2	do.	7 feet by 10 feet	Conc. closed	-	-

LEAD AND ZINC ORES

20	No. Idaho	2,220	Various	1	Ball	8 feet by 36 inches	Low-mid. closed	18	66
	West			1	do.	6 feet by 22 inches	Hi-mid. open	26	79
				2	do.	do.	Hi-mid. closed	25	76
	South			1	Ball	8 feet by 36 inches	Pri. part. cl.	-	-
				1	do.	do.	Sec. closed	-	-
				2	do.	6 feet by 22 inches	-	-	-
	Sweeney			1	Ball	8 feet by 36 inches	Closed	21	74
	Crescent			1	Ball	6 feet by 22 inches	-	26	79
				1	do.	4 feet by 4 feet	-	28	72
	Deadwood			1	Ball	6 feet by 22 inches	Closed	26	79
21	Chief Cons.	165	D	1	Ball	8 feet by 48 inches	Closed	20	72
22	Netta	² 600	Modified E	1	Ball	6 feet by 10 feet	Closed	20	61
23	White Bird	² 300	Modified E	1	Ball	6 feet by 36 inches	Closed	24	73
24	Black Hawk	170	D	1	Ball	6 feet by 6 feet	Closed	25	78
25	Mascot	1,900	Cr + A	1	Rod	7 feet by 10 feet	Pri. open	17	55
				1	Ball	5½ feet by 22 inches	Sec. open	23	67

²Tons per 10 hours.

Table 1.- Operating data of fine grinding mills - Continued
(Section A - Continued)

LEAD AND ZINC ORES - Continued

No.	Property	Plant rated tons per day	Type of flow sheet	Grinding mills			Circuit	Speed	
				No.	Kind	Size		R.p.m.	Per- cent crit- ical
1	2	3	4	5	6	7	8	9	10
26	Tybo	320	D	2	Ball	8 feet by 48 inches	Closed	19	68
27	Hughesville	400	Dr	1	Rod	6½ feet by 12 feet	Closed	18	57
28	Midvale	1,000	Dr + E	3	Rod	5 feet by 10 feet	Pri. closed	18	49
				3	Ball	5 feet by 10 feet	Sec. closed	27	75
29	Ruby	250	F	1	Ball	8 feet by 36 inches	Closed	16	58
30	Balmat	500	Dr	1	Rod	6½ feet by 12 feet	Closed	18	55
31	Morning	1,200	D' + D, and E	5	Ball	8 feet by 36 inches	Pr. part. cl.	21	75
				2	do.	do.	Sec. closed	21	75
				2	do.	8 feet by 22 inches	Mid. closed	23	82
32	Page	300	D' + A + E	1	Ball	6 feet by 4 feet	Pri. part. cl.	25	78
				1	do.	6 feet by 16 inches	Sec. open	28	85
				1	do.	8 feet by 36 inches	Tert. closed	22	78
33	Hecla	900	Modified E and D	1	Ball	5 feet by 6 feet	Closed	-	-
				1	do.	6 feet by 22 inches	do.	-	-
				3	do.	8 feet by 36 inches	Retr. closed	21	75
34	Pecos	600	D	2	Ball	7 feet by 5 feet	Closed	22	78
35	Flat River	5,000	Modified Er + conc. + modi- fied Er	6	Rod	6½ feet x 12 feet	Pri. closed	17	54
				1	do.	5 feet by 9 feet	Mid. closed	21	58
36	Potosi	2,400	Dr	4	Rod	7 feet x 12 feet	closed	16	54
				2	do.	5 feet x 12 feet	do.	22	60

Table 1.- Operating data of fine grinding mills - Continued
(Section A - Concluded)

GOLD ORES

No.	Property	Plant rated tons per day	Type of flow sheet	Grinding mills			Circuit	Speed	
				No.	Kind	Size		R.p.m.	Per- cent crit- ical
1	2	3	4	5	6	7	8	9	10
37	Alaska-Juneau	11,000	D, and (C or D) + D	11	Ball	8 feet by 6 feet	Pri. op. or cl.	22	80
				5	do.	6 feet by 12 feet	Sec. closed	25	80
38	Homestake	2,000	Fr + Bp + Ep	8	Rod	5 feet by 10 feet	Pri. closed	20	55
				2	Pebble	5 feet by 14 feet	Sec. open	-	-
				1	do.	do.	Tert. closed	-	-
				1	do.	6 feet by 72 inches	do.	-	-
39	Spring Hill	150	D	2	Ball	7 feet by 36 inches	Closed	23	76
40	Porcupine	25	D	1	Ball	4½ feet by 16 inches	Closed	31	83
41	Kirkland Lake	150	A + modi- fied E	1	Ball	8 feet by 30 inches	Pri. open	16	58
				1	do.	5½ feet by 16 feet	Sec. closed	28	82
42	Coniaurum	500	Modified E	1	Ball	5 feet by 16 feet	Closed	28	77

MISCELLANEOUS ORES

43	Minpro	² 75	Dp	1	Pebble	8 feet by 48 inches	Closed	28	98
				2	do.	8 feet by 36 inches	do.	28	98
				1	do.	4 feet by 16 feet	do.	-	-
44	Questa	50	D	1	Ball	5 feet by 4 feet	Closed	27	75

²Tons per 10 hours.

Table 1.- Operating data of fine grinding mills - Continued

(Section B)

COPPER ORES

No.	Property	Grinding media				Pulp, per- cent solids	Circu- lating load, per- cent	Power		Mill closed or followed by
		Pounds	Per- cent mill vol- ume	Size	Kind			Horse- power of each	All, per- cent plant total	
		11	12	13	14	15	16	17	18	19
1	Hayden	32,000	32	2 inches	c.s.	80	264	140	30	14 Dorr, 8 by 28 feet.
2	Cananea	44,000	41	3½ inches by 12 feet	-	-	-	112	18	Dorr, 4 by 12 feet and bowl 6 by 28 by 12 feet
		18,000	31	2½ inches	-	68	-	91	31	4 Drag.
		30,000	50	2½ inches	-	68	700	116		2 Dorr, 6 by 23 feet.
3	Harmony	27,000	52	3½ inches	c.i.	72	-	89	50	Dorr, 4½ by 16 feet and flota- tion.
4	Magma	9,000	25	5 inches	f.s.	62	45	34	12	3 Callow screen and tables.
		18,000	38	2 inches	c.i.	67	100	28	10	3 Esperanza, 2½ by 38 feet.
5	United Verde	66,000	40	3¼ inches	c.i.	81	400	¹ 300	60	4 Dorr, 8 by 30 feet.
		66,000	40	2½ inches	do.	78	100	¹ 268		Bowl 8 by 33 by 22 feet.
6	C. & H. recl.	-	-	-	peb.	-	-	-	-	Tables.
7	Nacozari	36,000	30	3 inches by 12 feet	-	78	0	131	32	3 Bowl, 6 by 20 by 13 feet.
		36,000	30	2½ inches by 12 feet	-	73	170	124	31	6 Dorr, 6 by 23 feet.
8	C. & H. congl.	-	-	4 inches	peb.	25	-	-	-	-
9	Hurley	42,000	34	3 inches by 9 feet		65	0	155	15	9 Bowl, 8 by 27 by 15 feet.
		36,000	35	2 inches	c.i.	70	-	162	31	18 Dorr, 8 by 27 feet.
10	Copper Queen	38,000	31	3 inches by 12 feet		74	0	141	35	8 Bowl, 6 by 28 by 15 feet.
		-	-	3½ inches	c.s.	75	-	121	30	8 Dorr, 6 by 23 feet.

¹Estimated from other data as explained in the text.

Table 1.- Operating data of fine grinding mills - Continued
 (Section B - Continued)
 COPPER ORES - Continued

No.	Property	Grinding media				Pulp, per- cent solids	Circu- lating load, per- cent	Power		Mill closed or followed by
		Pounds	Per- cent mill vol- ume	Size	Kind			Horse- power of each	All, per- cent plant total	
		11	12	13	14	15	16	17	18	19
11	Morenci	40,000	39	3 inches	c.i.	-	125	-		3 Dorr, 8 by 26 feet.
		66,000	36	3 inches	c.i.	-	125	-	53	3 do.
		36,000	30	2 inches	c.i.	-	-	-		5 Bowl, 8 by 31 by 18 feet
12	Old Dominion	13,000	30	2½ inches	c.i.	50	0	80		Bowl, 8 by 23 by 18 feet.
		-	-	2 inches	c.i.	60	360	¹ 70	56	Dorr, 6 by 20 feet and flota- tion.
		-	-	2 inches	c.i.	70	360	¹ 80		Dorr, 8 by 20 feet and flota- tion.
		41,000	34	3 inches by 12 feet		70	300	130		Dorr, 8 by 27 feet.
13	Arthur and Magna	42,000	41	3-3/8 inches	c.i.	-	0	-		
		42,000	41	2 inches	c.i.	-	-	-	45	39 Drag, 12 by 16 feet.
		42,000	41	2-3/8 inches	c.i.	-	-	-		18 Drag, 8 by 14 feet.
		42,000	41	2 inches	c.i.	-	-	-		36 Drag, 6 by 18 feet and 36, 8 feet by 18 feet.
14	Verde Central	-	-	3 inches	c.i.	-	-	-	53	2 Dorr, 6 by 26 feet.
15	Matahambre	9,000	25	4½ inches	f.s.	83	110	-	-	2 Trommel, 4 by 13 feet.
		9,000	20	3 inches	f.s.	80	150	-	-	4 Dorr, 8 by 18 feet.
		7,500	34	3 inches	f.s.	80	-	-	-	Dorr, 3 by 18 feet.
16	Engels	24,000	36	5 inches	c.s.	80	0	147		3 Dorr, 6 by 24 feet.
		25,000	28	5 inches	c.s.	83	33	268		1 do.
		28,000	32	2½ inches	c.s.	76	315	160	57	2 do.
		15,000	20	2½ inches	c.s.	76	340	129		1 do.
		24,000	36	2½ inches	c.s.	76	340	100		1 do.

¹Estimated from other data as explained in the text.

Table 1.- Operating data of fine grinding mills - Continued

(Section B - Continued)

COPPER ORES - Continued

No.	Property	Grinding media				Pulp, per- cent solids	Circu- lating load, per- cent	Power		Mill closed or followed by
		Pounds	Per- cent mill vol- ume	Size	Kind			Horse- power of each	All, per- cent plant total	
		11	12	13	14	15	16	17	18	19
17	Walker	18,000	30	4 inches	f.s.	78	145	¹ 157	59	3 Dorr, 6 by 18 feet.
		26,000	31	4 inches	f.s.	78	145	¹ 220		Dorr, 6 by 23 feet.
18	Miami	-	-	-	-	-	-	-		12 Dorr, 6 by 19 feet.
		-	-	-	-	-	-	-		12 do.
		-	-	-	-	-	-	-	52	Bowl, 20 by 42 by 28 feet.
		-	-	-	-	-	-	-		Bowl, 6 by 23 by 15 feet.
		-	-	-	-	-	-	-		Bowl, 6 by 23 by 15 feet.
19	Britannia	14,000	17	3½ inches	f.s.	70	0	-		-
		14,000	26	3½ inches	f.s.	70	-	-	59	18 Dorr, 3 by 20 feet.
		-	-	1½ inches	f.s.	-	-	-		Drag and bowl.

LEAD AND ZINC ORES

20	No. Idaho	-	-	3 inches	c.i.	70	334	125	-	Dorr, 6 feet.
	West	-	-	3 inches	c.i.	51	0	-	-	-
		-	-	2½ inches	c.i.	76	860	-	-	Dorr, 6 feet.
	South	-	-	3 inches	c.i.	53	100	158	-	Dorr, 6 feet.
		-	-	3 inches	c.i.	-	400	125	-	do.
		-	-	3 inches	c.i.	74	-	45	-	-
	Sweeney	-	-	3 inches	c.i.	-	266	134	-	Esperanza, 5 by 18 feet.
	Crescent	-	-	3 inches	c.i.	-	300	48	-	Dorr, 2½ feet.
21		-	-	2 inches	c.i.	-	300	-	-	
	Deadwood	-	-	3 inches	c.i.	-	500	45	-	Dorr, 4½ feet.
	Chief Cons.	-	-	5 inches	f.s.	75	335	180	38	Dorr, 6 by 20 feet.
22	Netta	30,000	-	3 inches	f.s.	-	-	-	-	Drag and tables.
23	White Bird	16,000	45	3 inches	f.s.	70	-	-	16	Screen and tables.
24	Black Hawk	13,000	27	4 inches	f.s.	72	300	-	-	Dorr, 6 by 18 feet.

¹Estimated from other data as explained in the text.

Table 1.- Operating data of fine grinding mills - Continued

(Section B - Continued)

LEAD AND ZINC ORES - Continued

No.	Property	Grinding media				Pulp, per- cent solids	Circu- lating load, per- cent	Power		Mill closed or followed by
		Pounds	Per- cent mill vol- ume	Size	Kind			Horse- power of each	All, per- cent plant total	
		11	12	13	14	15	16	17	18	19
25	Mascot	55,000	43	3 inches by 10 feet	-	-	0	146	-	Dorr, 8 by 25 feet.
		40,000	31	2½ inches	-	-	0	158	-	-
26	Tybo	36,000	46	3 inches	c.s.	76	400	139	65	2 Dorr, 6 by 25 feet.
27	Hughesville	-	-	-	-	-	300	195	36	Dorr, 8 by 30 feet.
28	Midvale	20,000	34	2 inches by 10 feet	-	78	57	162	35	3 Akins, 54 inches
		15,000	31	2½ inches	c.i.	78	-	100	-	3 Akins, 54 inches.
29	Ruby	32,000	48	3 inches	c.i.	-	-	135	35	Drag.
30	Balmat	-	-	2½ inches by 12 feet	-	-	207	181	21	Dorr, 8 by 30 feet.
31	Morning	32,000	48	4 inches	c.i.	-	-	-	-	5 Esperanza, 8 by 21 feet.
		26,000	39	3 inches	c.i.	-	-	-	46	2 do.
		-	-	1½ inches	f.s.	-	-	-	-	Thickener, 18 feet.
32	Page	-	-	-	-	-	-	-	-	Dorr 4 feet.
		-	-	-	-	-	-	-	-	-
		-	-	-	-	-	-	-	-	Esperanza.
33	Hecla	-	-	-	-	-	-	-	-	Drag, 6 by 14 feet.
		-	-	-	-	-	-	-	-	-
		-	-	-	-	-	300	-	-	3 Dorr, 5 by 18 feet.
34	Pecos	-	-	4½ inches	f.s.	-	400	151	36	2 Dorr, 6 by 23 feet.
35	Flat River	55,000	45	2 inches by 12	-	66	-	190	24	Screens and tables.
		28,000	50	2 inches by 9 feet	-	66	-	95	-	Flotation.

¹Estimated from other data as explained in the text.

Table 1.- Operating data of fine grinding mills - Concluded

(Section B - Concluded)

LEAD AND ZINC ORES - Concluded

No.	Property	Grinding media				Pulp, per-cent solids	Circu-lating load, per-cent	Power		Mill closed or followed by
		Pounds	Per-cent mill vol-ume	Size	Kind			Horse-power of each	All, per-cent plant total	
		11	12	13	14	15	16	17	18	19
36	Potosi	-	-	3 inches	-	-	-	¹ 110		Dorr.
				by 12 feet					40	
		-	-	do.	-	-	-	¹ 47		do.

GOLD ORES

37	Alaska-Juneau	20,000	22	6 inches	f.s.	40	140	187	-	Trunnion trommel
		20,000	21	5 inches	c.s.	40	90	113	-	do.
38	Homestake	23,500	40	3 inches	-	-	-	76	-	8 Dorr, 4 by 22 feet.
				by 10 feet						
		-	-	2½ inches	peb.	74	0	-	-	
		-	-	2½ inches	peb.	-	-	-	-	Dorr, 4 by 22 feet.
		-	-	2½ inches	peb.	-	-	-	-	
39	Spring Hill	-	-	4 inches	c.i.	55	500	-	56	2 Dorr, 4 by 14 feet.
40	Porcupine	4,000	42	4 inches	f.s.	34	125	-	44	Amalgamation and Dorr, 4 by 15 feet.
41	Kirkland Lake	30,000	50	5 inches	f.s.	75	0	-		
		15,000	16	1½ inches	f.s.	73	115	-	70	Dorr, 6 by 20 feet and bowl.
42	Coniaurum	-	-	2 inches	f.s.	70	-	-	-	Dorr, 6 by 30 feet and bowl.

MISCELLANEOUS ORES

43	Minpro	12,800	42	4 inches	peb.	-	-	-		Air separator and screens.
		11,600	45	4 inches	peb.	-	-	-	87	
		6,000	33	2 inches	peb.	-	-	-		
44	Questa	4,200	22	4 inches	f.s.	80	280	43	43	Dorr, 3 by 18 feet.

¹Estimated from other data as explained in the text.

Table 2.-- Products, reagents in mill feed, and costs of fine grinding mills
(Section A)

COPPER ORES															
No.	Property and year	Circuit	New feed per hour, tons	Skeleton-size analyses of				Reagents added in mill	Cost per ton concentrated		Distribution of grinding costs, percent			Iron wear per ton milled, pounds	
				Feed		Finished product			Total	For grinding	Labor	Supplies	Power	Balls or rods	Liners
				Percent	Mesh	Percent	Mesh								
1	2	3	4	5	6	7	8	9	10	11	12	13	14		
1	Hayden (1928)	Ball	241	98 10 200	97 65 200	Milk of lime	\$0.399	\$0.078	19	35	46	1.00	0.07		
2	Cananea (6 mo., 1929)	Pri. rod	102	95 1 inch 200	99 10 200	Dry lime		.062	12	40	48	.60	.04		
3	Harmony (3½ mo. 1929)	Sec. ball	8	98 10 200	98 48 200		.617	.116	3	44	53	1.36	.14		
4	Magna (6 mo., 1929)	Pri. ball	33	98 1 inch 200	97 14 200	Dry lime		.139	21	63	16	1.25	.27		
5	United Verde (Sept. 1929)	Sec. ball	-	98 14 200	99 48 200	Aerofloat	.934	.10	24	57	19	1.75	.19		
6	C. & H. recl. (1929)	Pebble	128												
7	Nacozari (1928-1929)	Pri. rod	108	95 1½ inch 200	99 8 200	Dry lime	.495	.172	7	39	54	-	-		
		Sec. rod		98 8 200	94 48 200		.513	.117	11	50	39	1.19	.39		

Table 2.- Products, reagents in mill feed, and costs of fine grinding mills - Continued

(Section A - Continued)														
COPPER ORES - Continued														
No.	Property and year	Circuit	New feed per hour, tons	Analyses of 100 lb. samples of			Cost per ton of concentrates	Distribution of grinding costs, percent		Labor Supplies	Power	Iron wear per ton milled, pounds		
				Feed	Percent	Mesh		Total	For grinding					
													Finished product	added in mill
2		3	4	5	6	7	.8	.9	10	11	12	13	14	
8	C. & H. congl. (1929) Hurley	Pebble	170	-	-		.533	.081	8	31	61	5.0*		
		Pri. rod	560	94	4	10		-	-	-	-	-	.40	
				15	200	200 Milk of lime	.031	-	-	-	-	-	-	
		Sec. ball	-	99	10	65		-	-	-	-	-	1.46	-
10	Copper Queen (1929)	Pri. rod	178	99	1 1/2 inch	20	.493	.111	17	46	37	1.34	.04	
			10	13	200	200 Dry lime		-	-	-	-	-	-	
		Sec. ball	-	98	20	65		.083	14	44	42	1.41	.08	
				13	200	200		.194	-	-	-	-	-	
11	Morenoi (April 1930)	Pri. ball	216	33	48	48 Milk of lime		.074	-	-	-	1.19	.21	
				19	200	200		.519	-	-	-	-	-	
		Sec. ball	-	98	-	65 Potassium ethyl xanthate		.051	-	-	-	-	.69	.03
				72	200	200		.125	-	-	-	-	1.32	.22
12	Old Dominion	Pri. ball	52	95	3	35 Milk of lime		.094	-	-	-	1.12	.03	
				9	200	200		.699	.094	11	20	22	1.12	.03
		Sec. ball	-	70	35	48 Milk of lime		.093	-	-	-	-	1.28	.08
				8	200	200 Oils		.187	.187	10	22	22	1.28	.08
		Rod	11	97	3/4 inch	48 Milk of lime		.187	10	22	22	1.28	.08	
				11	200	200 Oils		.187	10	22	22	1.28	.08	

* Pebbles.

Table 2.- Products, reagents in mill feed, and costs of fine grinding mills - Continued
(Section A - Continued)

COPPER ORES - Concluded													
No.	Property and year	Circuit	New feed per hour, tons	Skelloion-size analysis of			Reagents added in mill	Cost per ton concentrated			Distribution of grinding costs, percent		
				Percent	Mesh	Finished product		Total	For grinding	Labor	Supplies	Power	Iron wear per ton milled, pounds
1	2	3	4	5	6	7	8	9	10	11	12	13	14
13	Arthur and Magna (1 mo., 1929)	Pri. ball	-	99	10	-	-						
		Sec. ball	-	26	200	-	-	.302	.092	12	33	55	-
14	Verde Central (June 1929 to Aug. 1930)	Ball	17	-	1/4 inch	96	48 Dry lime	1.26	.37	21	33	46	-
15	Matahambre (1930)	Pri. ball	54	98	4	99	4 Dry lime		.127	-	-	-	-
		Sec. ball	-	3	200	29	200 NaCN	.747					
				98	14	99	35 Pine oil		.136	-	-	-	.54
				25	200	44	200		.263				
16	Engels (1929)	Pri. ball	72	98	1 inch	98	8 Dry lime		.153	17	48	35	1.08
				4	200	30	200 Fuel oil						.21
								.652					
		Sec. ball	-	97	3	99	48		.094	22	41	37	.72
				12	200	65	200		.247				.10
17	Walker	Ball	72	95	1 inch	98	30 Slaked lime	-	-	-	-	-	2.07
				1	200	44	200 Pine oil						.56
18	Miami (1930)	Ball	-	-	3	97	48	.297	.091	18	46	36	-
19	Britannia (July 1931)	Pri. ball	-	96	4	-	Milk of lime						
		Sec. ball	-	-	-	97	28	.177	.046	23	45	32	2.5
				40	200	40	200						.18

Table 2.- Products, reagents in mill feed, and costs of fine grinding mills - Continued
(Section A - Continued)

LEAD AND ZINC ORES

LEAD AND ZINC ORES																
No.	Property and year	Circuit	New feed per hour, tons	Skeleton-size analyses of				Reagents added in mill	Cost per ton concentrated		Distribution of grinding costs, percent			Iron wear per ton milled, pounds		
				Feed		Finished product	Mesh		Percent	Total	For grinding	Labor	Supplies	Power	Balls or rods	Liners
				Percent	Mesh											
1	2	3	4	5	6	7										
20	N. Idaho (1929)	Ball	-	98	3	100	Cresylic, NaCN	1.270	.292	7	61	32	3.0	-		
21	Chief Cons. (1929)	Ball	7	93	1 inch	65	Aero. ZnSO ₄	3.217	.455	17	40	43	3.42	.71		
24	Black Hawk (2 mo., 1929)	Ball	7	95	1/4 inch	100	Soda ash	1.55	.355	-	-	-	1.5	-		
25	Mascot	Pri. rod	29	6	200	200	Thio.	-	-	-	-	-	-	-		
26	Tybo (March 1930)	Sec. ball	-	-	-	-	-	-	.053	-	-	-	-	-		
27	Hughesville	Ball	14	95	1 1/4 inch	65	Soda ash	1.282	.445	12	41	47	2.7	-		
28	Midvale	Rod	13	99	1 inch	200	NaCN, ZnSO ₄	-	-	-	-	-	1.99	-		
		Pri. rod	45	94	1/2 inch	28	-	-	-	-	-	-	1.20	.21		
		Sec. ball	-	13	200	200	-	-	-	-	-	-	1.10	.21		
29	Ruby (Oct. 1929 to April 1930)	Ball	-	99	28	48	-	-	-	9	41	50	3.19	-		
30	Balmat	Rod	24	92	3/4 inch	35	Soda ash, NaCN	2.202	.252	-	-	-	2.2	-		
31	Morning (1929)	Pri. ball	-	5	100	200	Thio., ZnSO ₄	-	-	-	-	-	-	-		
		Sec. ball	-	99	65	200	Thio.	1.012	.193	12	45	43	2.19	.4		

Table 2.- Products, reagents in mill feed, and costs of fine grinding mills - Continued
(Section A - Continued)

LEAD AND ZINC ORES - Continued

No.	Property and year	Circuit	New feed per hour, tons	Skeleton-size analyses of			Reagents added in mill	Cost per ton concentrated		Distribution of grinding costs, percent			Iron wear per ton milled, pounds	
				Percent	Feed	Finished product		Total	For grinding	Labor	Supplies	Power	Balls	Liners
1	2	3	4		5	6	7	8	9	10	11	12	13	14
32	Page (1930)	Pri oall	-	10	1 inch 200	-	-							
		Sec. ball	-		-	-	-	.905	.411	17	51	32	-	-
		Tert. ball	-		-	99	-							
34	Pecos (1930)	Ball	22	99	1 1/2 inch 200	98	-	1.190	272	18	57	25	1.8	.3
35	Flat River	Pri. rod	105	99	6	99	-							
		Mid rod	-	97	10	98	-							
				11	200	82	-							
36	Potosi (July 1932)	Rod	96	95	1 1/2 inch 200	97	Cresylic	.504	.157	6	31	63	.54	-
				4	200	48	NaCN							

GOLD ORES

37	Alaska-Juneau (1928)	Pri. ball	165	98	2 1/2 inch 7	96	10						.93	.25
				5		27	200							
		Sec. ball	-	92	3/4 inch 14	96	10	326	.16	-	-	-	1.35	1.01
				9		30	150							
38	Homestake (1929)	Pri. rod	66	57	10	39	50		.084	22	55	23	1.30	.36
				3	200	49	200							
		Sec peb.	-	97	30			503						
				2	200									
		Tert peb.	-		-									
				95	50				.014	47	25	27	-	-
				40	200				.098					

Table 2.- Products, reagents in mill feed, and costs of fine grinding mills Concluded
(Section A - Concluded)

GOLD ORES - Continued														
No.	Property and year	Circuit	New feed per hour, tons	Skeleton-size analyses of			Reagents added in mill	Cost per ton concentrated		Distribution of grinding costs, Percent			Iron wear per ton milled, pounds	
				Feed	Finished product			Total	For grinding	Labor	Supplies	Power	Balls of rods	Liners
					Percent	Mesh								
1	2	3	4	5	6	7	8	9	10	11	12	13	14	
39	Spring Hill (July 1929 to April 1930)	Ball	8	93 1/2 inch	96 80	Pine oil	1.007	.290	35	25	40	1.3	-	
40	Porcupine	Ball	1	6 100	60 200	K-xan. Aero.								
41	Kirkland Lake (1930)	Pri. ball	-	75 1 inch	65 200	-	1.851	.771	56	21	23	1.5	-	
				69 100	98 200	Lime						1.96	.24	
				2	28									
		Sec. ball	-	-	95	300	1.389	.642	23	37	40			
42	Coniaurum	Ball	-	-	60	200 KCN	.939	.202	-	-	-	2.5	.26	
												1.8	-	
MISCELLANEOUS ORES														
43	Minpro (1929)	Pebble	-	-	-	-	1.45	.74	-	-	-	-	-	
44	Questa (10 mo., 1930)	Ball	2	99 1 1/2 inch	99 100	Pine oil	2.69	-	-	-	-	-	-	

Table 3.- Index of grinding mills

<u>Size</u>	<u>Number</u>	<u>Location and reference number</u>
<u>Diameter and length</u>	<u>in use</u>	
<u>Ball Mills:</u>		
10½ by 8 feet	6	Miami (18)
8½ by 12 feet	3	Morenci (11)
8 by 12 feet	3	United Verde (5)
8 by 6 feet	12	Alaska Juneau (37), Engels (16)
8 feet by 48 inches	3	Chief Cons. (21), Tybo (26)
8 feet by 36 inches	47	Old Dominion (12), Verde Central (14), Engels (16), Miami (18), No. Idaho (20), Ruby (29), Morning (31), Page (32), and Hecla (33)
8 feet by 30 inches	1	Kirkland Lake (41)
8 feet by 28 inches	6	Cananea (2)
8 feet by 22 inches	2	Morning (31)
7 by 10 feet	143	Hayden (1), Hurley (9), Morenci (11), Arthur and Magna (13), Britannia (19)
7 by 7 feet	1	Walker (17)
7 by 5 feet	5	Walker (17), Pecos (34)
7 feet by 36 inches	3	Harmony (3), Spring Hill (39)
6½ by 12 feet	5	Morenci (11)
6 by 12 feet	17	Alaska Juneau (37), Copper Queen (10), Engels (16), Miami (18)
6 by 11 feet	6	Britannia (19)
6 by 10 feet	2	Engels (16), Netta (22)
6 by 6 feet	5	Matahambre (15), Black Hawk (24)
6 by 4½ feet	7	Magma (4), Matahambre (15)
6 feet by 50 inches	5	Old Dominion (12)
6 by 4 feet	1	Page (32)
6 feet by 36 inches	1	White Bird (23)
6 feet by 22 inches	9	Miami (18), No. Idaho (20), Hecla (33)
6 feet by 16 inches	1	Page (32)
5½ by 22 feet	1	Mascot (25)
5½ by 16 feet	1	Kirkland Lake (41)
5½ by 9 feet	18	Britannia (19)
5 by 16 feet	1	Coniaurum (42)
5 by 10 feet	6	Magma (4), Midvale (28)
5 by 6 feet	1	Hecla (33)
5 by 5 feet	1	Matahambre (15)
5 by 4 feet	1	Questa (44)
4½ feet by 16 inches	1	Porcupine (40)
4 by 4 feet	1	No. Idaho (20)
<u>Rod Mills:</u>		
7 by 12 feet	4	Potosi (36)
7 by 10 feet	10	Hurley (9), Mascot (25)
6½ by 12 feet	8	Hughesville (27), Balmat (30), Flat River (35)
6 by 12 feet	24	Cananea (2), Nacozari (7), Copper Queen (10), Old Dominion (12)

Table 3.- Index of grinding mills - Continued

Size	Number	Location and reference number
<u>Diameter and length</u>	<u>in use</u>	
<u>Rod Mills (Continued:</u>		
5 by 12 feet	2	Potosi (36)
5 by 10 feet	11	Midvale (28), Homestake (38)
5 by 9 feet	1	Flat River (35)
<u>Pebble Mills:</u>		
8 feet by 72 inches	12	C. & H. congl. (8)
8 feet by 48 inches	1	Minpro (43)
8 feet by 36 inches	2	Minpro (43)
8 feet by 18 inches	64	C. & H. recl. (6)
6 feet by 72 inches	1	Homestake (38)
5 by 14 feet	3	Homestake (38)
4 by 16 feet	1	Minpro (43)
Total	470	

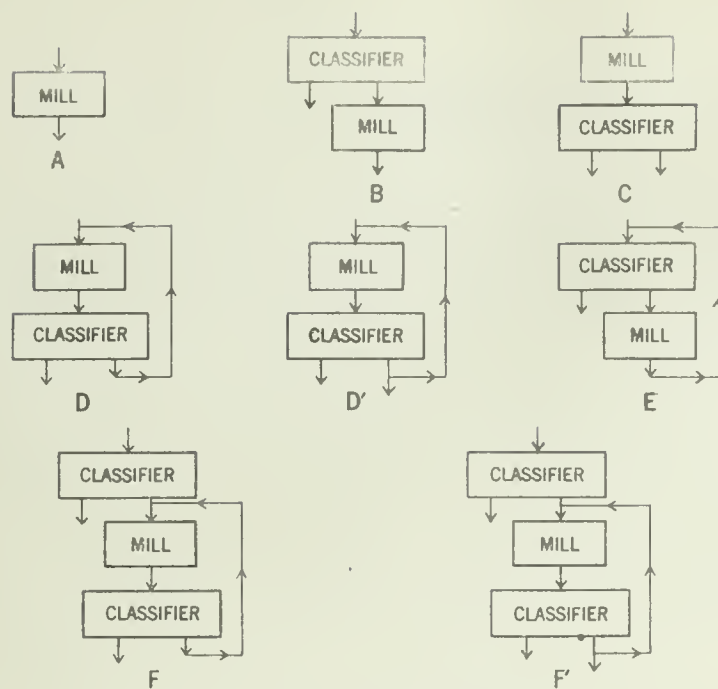


Figure 1.—Types of fine-grinding flow sheets.

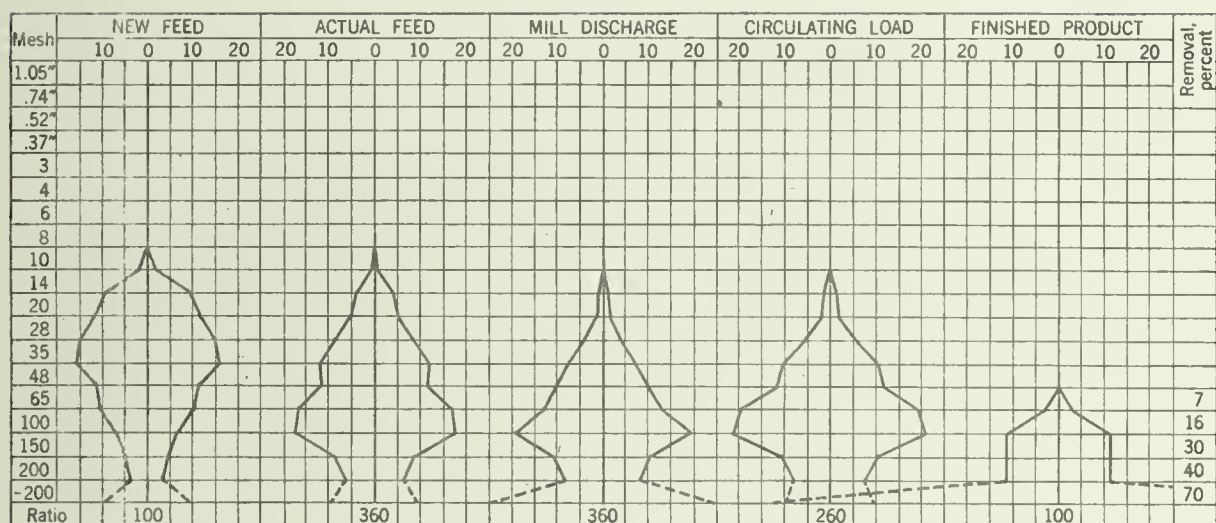


Figure 2.—Sizing diagrams of products of ball mill at Hayden (1).

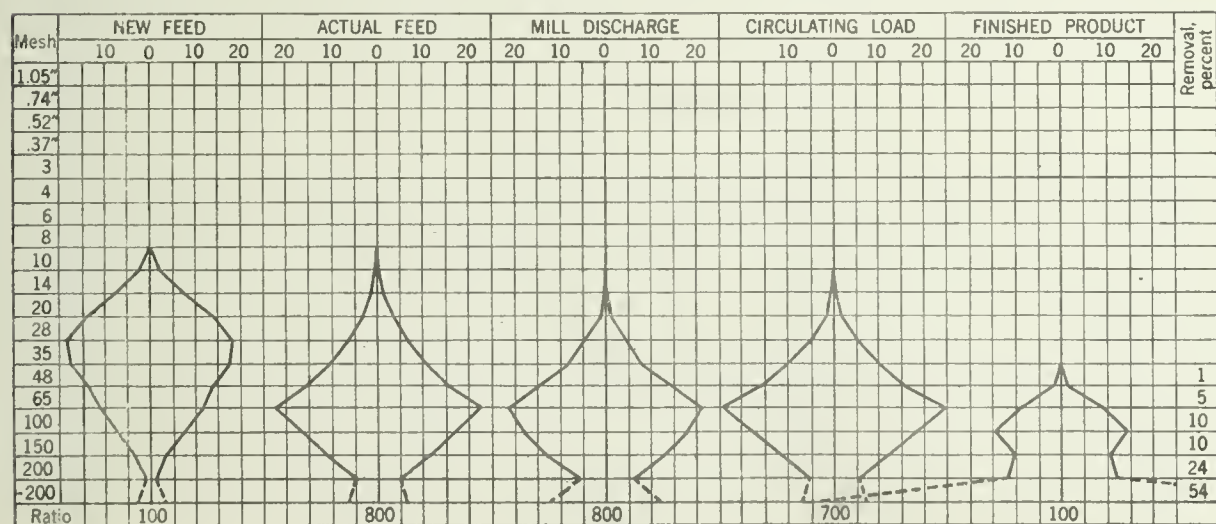


Figure 3.—Sizing diagrams of products of secondary ball mill at Cananea (2).

FLOW SHEETS

Ball, rod, or pebble mills are used either singly or in combinations with classifiers and with other mills. The simplest flow sheet is one in which the ore is ground sufficiently fine in one pass through a single machine. However, while certain particles are being ground to the required size in this open-circuit operation, others are being ground too fine; consequently, when other factors permit, better practice is to grind the ore in stages, removing the finished material intermittently from that which requires additional grinding. This additional grinding is accomplished by returning the unfinished sand either to the same mill (closed circuit) or to a secondary mill. In practice, flow sheets vary from simple open circuits to combinations of several closed-circuited mills in series. The separation of finished material from that in need of more grinding may be accomplished by mechanical classifiers, cones, screens, trommels, tables or other concentrating machinery, or air separators. To simplify the analyses of the flow sheets the following schematic combinations will be adopted (see fig. 1). By "mill" is meant ball mill, rod mill, or pebble mill, and by "classifier" is meant any of the machines used to separate the finished from the unfinished material in the grinding circuit.

Type A - Simple open-circuit mill.

Type B - Open circuit with classifier preceding mill which receives classifier oversize.

Type C - Open circuit with classifier following mill.

Type D - Simple closed circuit, with mill discharge feeding classifier and classifier oversize returning to mill.

Type D'- Simple partly closed circuit, like type D except only part of the classifier oversize returns to mill.

Type E - Inverted closed circuit, with classifier preceding mill which receives classifier oversize and returns mill discharge to the classifier.

Type F - Classified simple closed circuit, in which the mill receives classifier oversize as in type E, but discharges into a second classifier which is in simple closed circuit with the mill.

Type F'- Classified simple partly closed circuit, like type F except that only part of the oversize from the second classifier is returned to the ball mill.

In addition to these types, there are a few which, because of similarity, will be designated as modifications of the above general types. Subscripts r and p indicate that the mills are rod or pebble mills rather than ball mills. The types of flow sheets in the several fine-grinding sections are given in table 1, column 4.

OPERATION

Operating conditions which are discussed by the authors of the circulars are the speed of the mill; the amount, size, and kind of grinding media; the pulp density; the amount of circulating load when the circuit is closed; the type of lining; the size of the discharge opening; and the type and amount of the feed. Very often the conditions which require the most power result in the most grinding, but capacity alone is not always desirable. Ores may be overground with resulting losses in subsequent treatment, and power is often such an expensive item that it should be held at a minimum. Ball-mill operation should be considered with respect to local conditions, and consequently it is impossible to choose arbitrarily the conditions under which a ball mill should be operated for best results. Exigency may demand departure from a practice known to be scientifically sound.

Speed

To compare the speed of mills of different diameters, the speed as given in revolutions per minute has been translated into percent of the theoretical critical speed.⁵ This value for ball mills ranges from 53 percent [Morenci (11)] to 86 percent [Harmony (3)]; for rod mills it lies between 48 percent [Cananea (2)] and 66 percent [Hurley (9)]; and for pebble mills it ranges from 85 percent [C. & H. congl. (8)] to 98 percent [Minpro (43)]. See columns 9 and 10 of table 1. Slow speeds are sometimes used in the interest of economy [Kirkland Lake (41)] when additional grinding capacity is not needed, but in other instances [Harmony (3)] there cannot be the slightest decrease lest the grinding capacity drop.

Generally, no provision is made for varying the speed of a mill; and consequently variations in grinding are obtained by changing other operating conditions.

Grinding Media

Amount

Ordinarily, the amount of grinding media is known accurately only when the mill is started with a fresh charge or when an old charge is dumped and weighed. The loss due to wear is made up by periodic additions of balls, rods, or pebbles, at a rate which attempts to maintain the original volume. Figures on ball and rod wear will be found in table 2, column 13.

In table 1, column 12, the amount of grinding media is expressed in estimated percentages of the mill volume.⁶ Very seldom do mills operate with charges over 55 percent. Mill volumes as low as 17 percent [Kirkland Lake (41)] are reported, while in other mills [Magma (4)] the ball volume is increased only when additional capacity is required.

Comparative tests at Cananea (2) show that an increase in ball load from 9 to 15 tons (31 to 52 percent of mill volume) gave an increase of 67 percent in the minus 48-mesh material produced, and an increase in power of only 27 percent.

Size

The size of the grinding media added is established by the size of the coarsest particles in the feed. At Alaska Juneau (37) where 2½-inch material is present in the feed, balls as large as 6 inches in diameter are used. When grinding is done in several steps, large balls are used in the primary mills, and, generally, smaller ones in the following mills. For regrinding, balls as small as 1½-inch maximum are employed [Britannia (19); Morning (31)]. The average size of make-up balls is 2 to 4 inches. The tendency is to use balls of smaller diameter than those formerly used [Hayden (1), Harmony (3), Hurley (9)]; but at Magma (4) when the ball charge is unusually light, large balls are necessary to maintain tonnage.

Rod diameters, in general, are chosen according to the size of the feed [Nacozari (7)]. The tendency is toward a smaller diameter of rods [Flat River (35)] to avoid breakage which is attributed to large rods.

5 The critical speed is given by the formula: 54.19 divided by the square root of inside radius in feet. The inside diameter has been taken from trade catalogs, where possible, and otherwise is assumed to be 6 inches less than the nominal dimension. No correction for ball size has been made. The speed of conical mills is figured on the diameter of the cylindrical portion.

6 The inside diameters of the various mills are estimated as explained under "speed"; the lengths from manufacturers' data, or are assumed to be nominal.

Pebbles, because of their lesser density are larger than balls which might be used in their stead. Flint pebbles range in maximum size from $2\frac{1}{2}$ inches [Homestake (38)] to 4 inches [C. & H. (6) and (8), Minpro (43)].

Kind

Grinding balls are either cast or forged. Of the mills listed, 15 use cast balls, 12 use forged balls, and 4 use both. Forged steel balls have the longest life, but the cheapness of cast-iron or cast-semisteel balls often makes their use more economical [Harmony (3), Verde Central (14), Engels (16), and Midvale (28)].

Balls have replaced pebbles in several grinding circuits [Morenci (11), Old Dominion (12), Miami (18), Britannia (19), Hecla (33), and Kirkland Lake (41)], but in some instances where pebbles can be obtained at low cost, they compete successfully with balls [C. & H. (6) and (8)]. In certain installations, such as those for the grinding of feldspar [Minpro (43)], iron cannot be used because of contamination, and pebbles, together with silex or other nonmetallic lining, must be employed.

Rods are of high-carbon steel and are hot-sawed and machine-straightened. The analyses of rods [Cananea (2), Nacozari (7), Copper Queen (10), Homestake (38), Balmat (30), Flat River (35)] fall within the following limits: C, 0.6 to 1.0 percent; Mn, 0.5 to 0.8 percent; Cr, 0.2 to 0.5 percent; Si, 0.15 to 0.25 percent; P, 0.02 to 0.04 percent; and S, 0.03 to 0.05 percent.

No record is made of periodic inspections of the ball or rod charges.

Pulp Density

Pulp densities in ball mills vary from 34 percent solids [Porcupine (40)] to 83 percent solids [Matahambre (15)]. In most instances, the product of the fine-grinding circuit is classifier overflow which goes direct to flotation, but in a few mills [C. & H. (6) and (8), Arthur and Magna (13), Netta (22), White Bird (23), and Float River (35)], thickeners are ahead of flotation and the pulp density of the finished product need not be maintained at flotation density. At Harmony (3) a thick pulp, 68 to 76 percent solids, is maintained to reduce liner and ball wear and to increase capacity and the uniformity of the product; to facilitate control, a density indicator is kept in the discharge pulp. At Alaska Juneau (37) a thin pulp, 40 percent solids, is used to carry the ore through the mill rapidly, thus avoiding overgrinding.

Circulating Load

As may be seen from the tabulation of type of flow sheets in table 1, most of the plants do fine grinding in closed circuits. With the exception of Mascot (25) in which even the secondary ball mill is in open circuit, all secondary circuits are closed. In some cases the circulating load builds up to 400 or 500 percent [United Verde (5), N. Idaho (20), Tybo (26), Pecos (34), and Spring Hill (39)]. At Cananea (2) the circulating load was found to be 700 percent by a calculation similar to the one described in the Copper Queen (10) report; and in one circuit at the West concentrator of the N. Idaho mills (20) it reaches 860 percent.

The building up of sulphides in the classifiers in closed circuit with mills is noted in several plants [Cananea (2), Nacozari (7), Old Dominion (12), and Balmat (30)], and in one plant [Copper Queen (10)] the amount of circulating load fluctuates with the amount of sulphide in the ore.

At Alaska Juneau (37) in the primary and secondary circuits, and in the primary circuit of Matahambre (15), the circulating load is trommel oversize. In the primary circuits at Magma (4) and Flat River (35), and in the White Bird (23) circuit the circulating load is vibrating-screen oversize and table return. At Minpro (43) screens and air separators are used. At the remaining plants mechanical classifiers of the rake, drag, or screw type make a return sand which constitutes most of the circulating load, although flotation and table middlings are sometimes returned for regrinding.

Lining

The majority of the mills use cast iron or steel liners. However, sections of rail set in concrete are used at some plants, and rubber and stone linings are found occasionally. The C. & H. congl. (8) mills are lined with old steel rails, 60 pounds or heavier, imbedded in neat cement. At Hurley (9), 3½-inch lengths of rails are used in the side liners with cast iron end liners. The Arthur and Magna (13) mills have 4-inch lengths of 90-pound rail laid on end against the shell in concrete; and at Britannia (19) the lining is made of 7-inch sections of rail set on end in high-strength concrete. The rail sections in the last instance are hardened by heating and quenching in cold water before being cemented into the mill. The Britannia mills were designed originally as pebble mills, and to adapt them to ball loads an inner lining of 2-inch wooden lagging was placed between the shell and the rail-and-concrete lining.

At Cananea (2) a wood inner lining of 1½ inches is placed between the shell and the liner castings; and at Old Dominion (12) the inside diameter has been reduced by inserting 10-inch timbers under the cast-iron liners. The Old Dominion mills which were originally 8-foot by 36-inch conical mills now measure 6 feet by 50 inches. A 7-by 10-foot mill at Engels (16) was reduced with 6-inch timbers and cast-iron liners to a 6- by 10-foot size. At this concentrator, mills are stopped every 2 weeks, the liners are inspected, and thin liners are replaced. In spite of this precaution, however, the shell becomes worn by pulp flowing between the ends of the liner rows. Some experimental work has been done with a rubber lining between the shell and the regular lining, but no definite information as to the value of this inner lining has been reported.

At Flat River (35) liner sections are bolted into place with 1/8-inch shims between the liners and the shell, and then molten zinc is poured between and under the sections so as to form a continuous and cushioned all-metal lining in the mill.

At Homestake (38) rubber lining in pebble mills has given good service and standardizing its use is anticipated, but some cast-iron and Belgian silex linings are still being employed. At Minpro (43) silex is used and at Calumet & Hecla reclamation (6) silex and domestic quartzite comprise the lining material.

At Magma (4) feed-end liners are cast around steel reinforcing bars.

Flat River (35) reports that the liner wear is greatest at the discharge end of the rod mills, but in the Northern Idaho ball mills the feed-end liners wear out first. At Harmony (3) liner wear is reduced by using smaller balls and at Verde Central (14) by slow mill speed.

The amount of liner wear is given in table 2, column 14.

Discharge Openings

The size of the discharge openings varies. Generally, rod mills have larger discharges than ball mills, although the low pulp level of the grate type of ball mill is reputedly the equivalent of a large discharge, since it reduces the amount of the pulp at the discharge

end of the mill and creates a steeper pulp gradient within the mill. At Copper Queen (10) the rod mills (primary) have low-center discharges, whereas the ball mills (secondary), which are of similar design, have high-center discharges. At Nacozari (7) both primary and secondary grinders are rod mills, but they differ in the size of the discharge trunnions; the primary mills have 30-inch and the secondary have 18-inch openings. At both Nacozari (7) and Copper Queen (10) the mills with the larger discharge trunnions require more power; and at Cananea (2) low power is attributed in part to the high pulp level discharge.

Mills with grate discharges are used in at least five concentrators [Magma (4), Engles (16), Walker (17), Alaska Juneau (37), and Questa (44)]. Grate openings are from $\frac{1}{4}$ inch [Walker (17)] to 1 inch [Alaska Juneau (37)].

Allied with all these operating conditions is the type and amount of feed, which will be considered under "Products."

POWER

The gross horsepower consumption of the grinding mills is presented in table 1, column 17. In a few instances these values had to be estimated. Since the amount of dead load is not obtainable these values include not only the power used to impart action to the balls but also the motor loss and the power necessary to overcome friction in the bearings and drive mechanism. At Cananea (2) low power is attributed in part to roller bearings on the primary rod mills.

Power accounts for as much as 60 percent of the fine-grinding cost [C. & H. congl. (8); Potosi (36)]. Fine grinding is often the largest consumer of power in the entire concentrator, accounting for over half of the total power at 13 plants. See table 1, column 18.

PRODUCTS

Common practice is to feed the mills with the products of the fine crushers, even though this feed may contain material which is already of sufficient fineness. However, in concentrators with flow sheets of types B, E, and F classifiers precede the grinding mills and prepare the feed by removing sands and slimes which need no further grinding. Table 2, column 5, gives skeleton screen analyses of the feeds to the mills. The largest particles in these feeds range from $2\frac{1}{2}$ inches [Alaska Juneau (37)] to 10 mesh [Hayden (1), and Arthur and Magna (13)].

Operating conditions should be regulated to suit the particular feed, and it follows that once these conditions have been established the feed should be as uniform as practical. The report on Harmony (3) calls attention to the desirability of having ball-mill feed uniform in size, and this principle has led to large circulating loads and stage-grinding elsewhere. Two stages have been found more efficient than single stage at United Verde (5), and at Miami (18) three stages are used so that the grinding in each mill is confined to as small a range in reduction as possible. At Harmony (3) the tendency is to do more of the grinding in the fine-crushing department ahead of the ball mills and thereby make the ball-mill feed more uniform.

A steady rate of feed is likewise important to the best operation of a ball mill. Because of this, a surge or storage bin is often placed ahead of the fine-grinding department. At Engles (16) the feed rate to a primary mill is regulated largely by the sound of the mill. At Porcupine (40) the feed is cut off if the discharge becomes too coarse for amalgamation.

Finished Product

The size to which an ore is ground is determined by the degree of locking of gangue and mineral grains, the succeeding process, or the commercial demands. The finishing size ranges from 96 percent minus 10 mesh [Alaska Juneau (37)] to 96 percent minus 300 mesh [Kirkland Lake, (41)]. Skeleton screen analyses of finished products are shown in table 2, column 6.

Sizing diagrams of the products of several mills are shown in figures 2 to 9, inclusive. These diagrams are made in the manner described by Coghill⁷ and show the sizing analyses of (1) the new feed to the mill, (2) the actual feed, which is a weighted composite of the new feed and circulating load, (3) the mill discharge, (4) the circulating load, and (5) the finished product of the grinding circuit. The percentage of material retained on each sieve is plotted bilaterally to give a symmetrical diagram. The minus 200-mesh fractions are shown in dotted lines. Tall diagrams represent coarse products and short diagrams fine products. Wide areas indicate a concentration by weight at a particular size range. The numbers below the diagrams show the relative amounts of each product. The tonnages and other operating data for the circuits which are diagrammed may be found in the tables.

CLASSIFICATION

Fine grinding mills may be closed, or followed, by screens, concentrating machines, or classifiers of the thickener, drag, spiral, or rake type. In the concentrators reviewed, the latter are the more common; there are about 170 reciprocating rake classifiers in the ball- and rod-mill circuits alone. The number, size, and kind of classifiers are given in table 1, column 19.

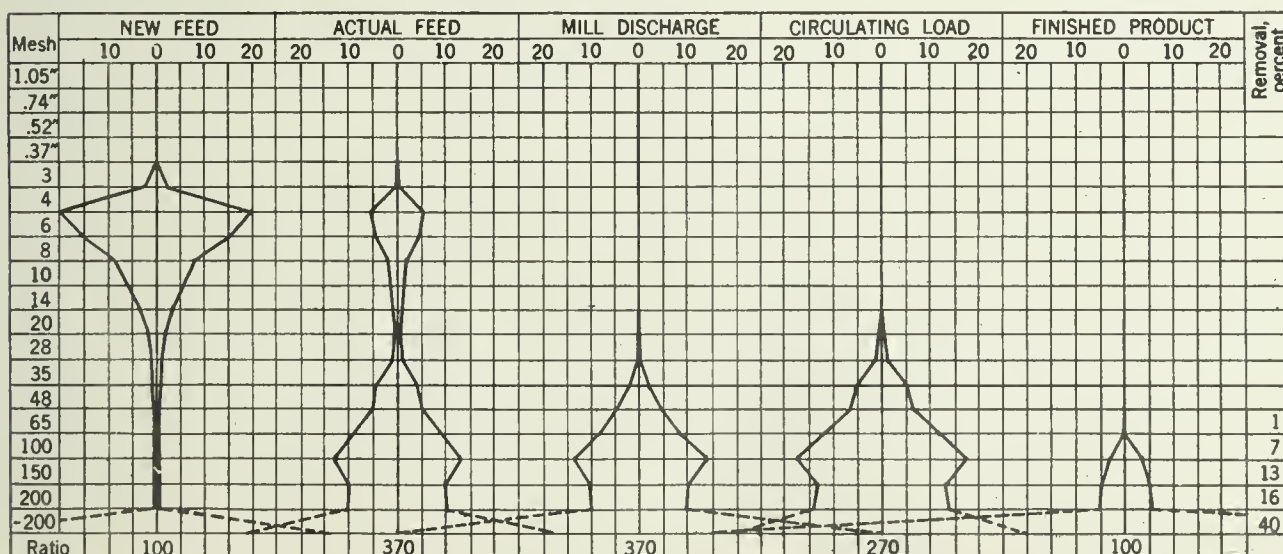
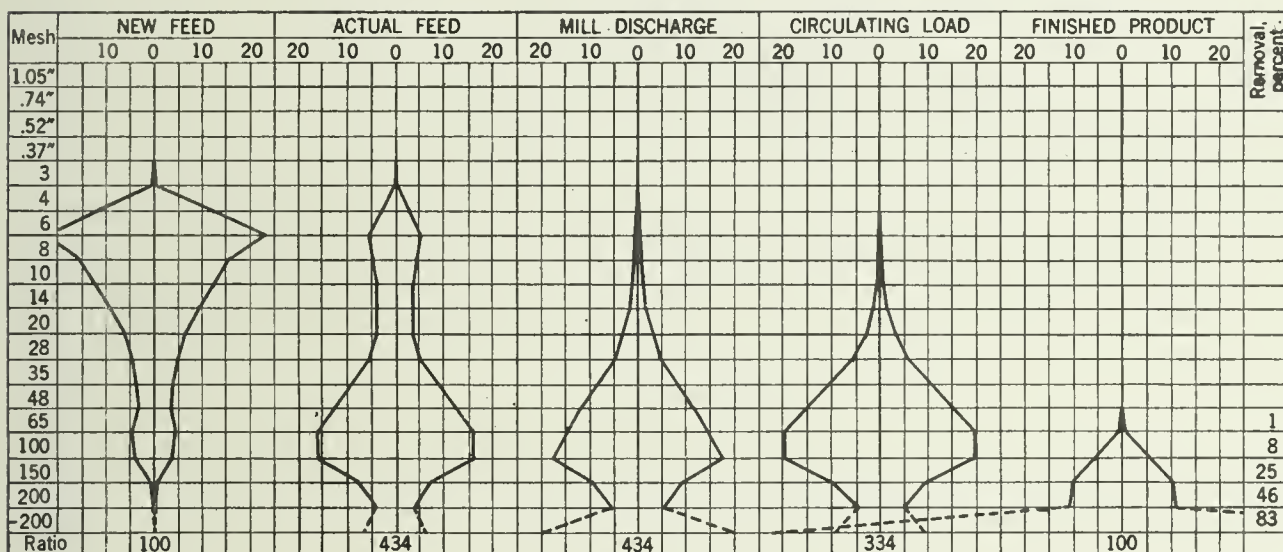
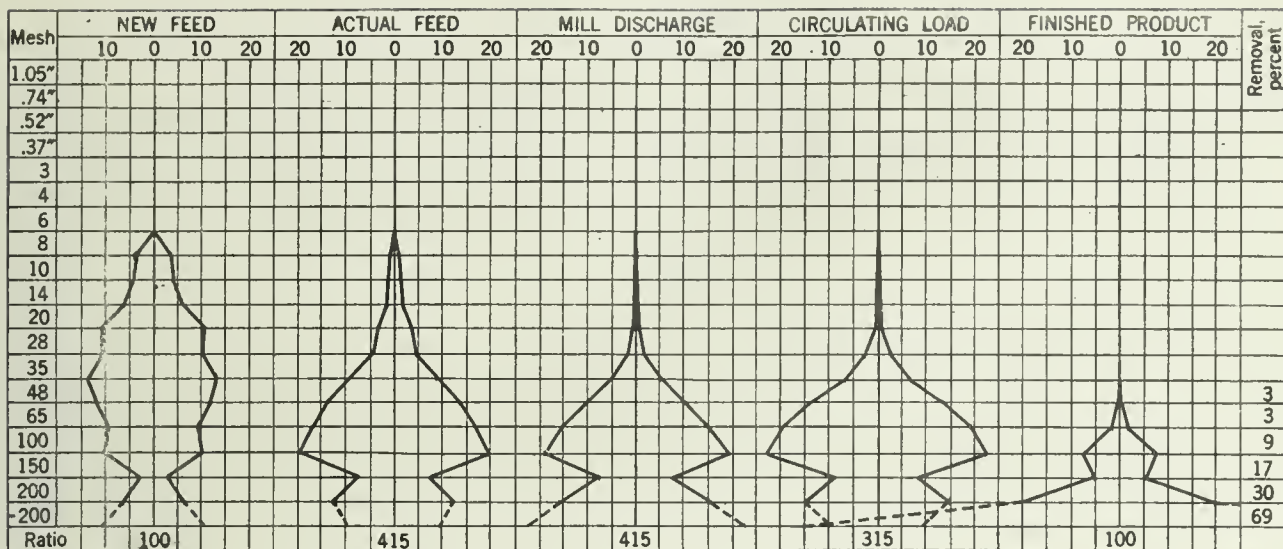
The size of the overflow and the degree of removal of the fines depend on the amount of material handled, the amount of water in the overflow, washing, slope, and rake speed. When fine overflows (say 100 mesh) are desired, the slope is as flat as 2 inches per foot, and the rake speed is about 12 strokes per minute [Porcupine (40), Kirkland Lake (41), and Questa (44)]; but for overflows containing 35- to 48-mesh material, slopes up to 4 inches per foot and speeds up to 35 strokes per minute are used [Cananea (2), Harmony (3), and Homestake (38)].

An average from 18 concentrators showed that the capacity of a rake classifier may be considered as 2 tons of solids per hour per linear foot of overflow lip.

The efficiency of the classification in several circuits is shown in the diagrams in figures 2 to 9 inclusive. The numbers under "removal, percent" are approximations of the amounts of the various size-increments of the mill discharge which were removed in the finished product. These numbers would be "100" for perfect classification at that size beyond which grinding is no longer necessary. However, the presence of heavy sulphides and locked or middling grains makes the finishing size indeterminable, so that no exact evaluation of the efficiency of the classifiers can be made without study and additional data.

At Cananea (2) and Nacozari (7) back-wash water is used to remove slimes from the classifier, but at Copper Queen (10) this practice was not found to be advantageous to the entire grinding circuit. At Harmony (3) and at Magma (4) water is added to the ball-mill discharge. At Questa (44) an indicating hydrometer is kept in the classifier overflow.

⁷ Coghill, Will H., Evaluating Grinding Efficiency by Graphical Methods: Eng. and Min. Jour., vol. 126, Dec. 15, 1928, pp. 934-8.



REAGENTS

Reagents are added to ball mills as shown in table 2, column 7. Of these, lime is the most common. It is generally added as a conditioning agent for some subsequent treatment, but at Nacozari (7) it serves also to prevent liner wear due to electrolysis.

COSTS

Power for fine grinding is not only the largest item in power consumption in many concentrators, but also, in some instances [Alaska Juneau (37), Minpro (43)] it accounts for half of the total milling cost. In other plants it amounts to 10 to 45 percent of the total. Fine-grinding costs as low as \$0.05 per ton have been obtained at Britannia (19) under very favorable conditions. Grinding costs and their distributions, together with the total milling costs, are given in table 2, columns 8 to 12, inclusive. These figures are per ton of ore concentrated, not necessarily per ton of ore ground.

Of the factors contributing to the cost of fine grinding, power is usually the most important. Power generated for 0.2 cent per kilowatt-hour, and locally manufactured balls and liners have made possible the phenomenally low cost mentioned at Britannia (19). The average recorded cost of power is about 1 cent per kilowatt-hour.

Supplies are chiefly balls and liners, and table 2, columns 13 and 14, shows the iron wear in pounds per ton of ore concentrated. When balls or liners are made locally, scrap can be utilized and the wastage from worn liners, broken rods, and rejected balls can be minimized [Cananea (2), Nacozari (7), Morenci (11), Old Dominion (12), Arthur and Magna (13), Britannia (19), Alaska Juneau (37), and Spring Hill (39)].

At Hughesville (27) wear on the lip of the feed scoops has been reduced by applying stellite with an acetylene welding torch.

SUMMARY

A review has been made of the fine-grinding practice in 49 concentrating plants in the United States, Alaska, Canada, Mexico, and Cuba; and the operation of 470 ball, rod, and pebble mills has been set forth. The methods of operation, grinding capacities, and milling costs have been assembled and tabulated to show the present practice and to facilitate comparisons between various mills. Considerable diversity in the details of operation and in the grinding accomplished is apparent; and certain trends in fine-grinding practice, and also the variables of operation have been indicated.

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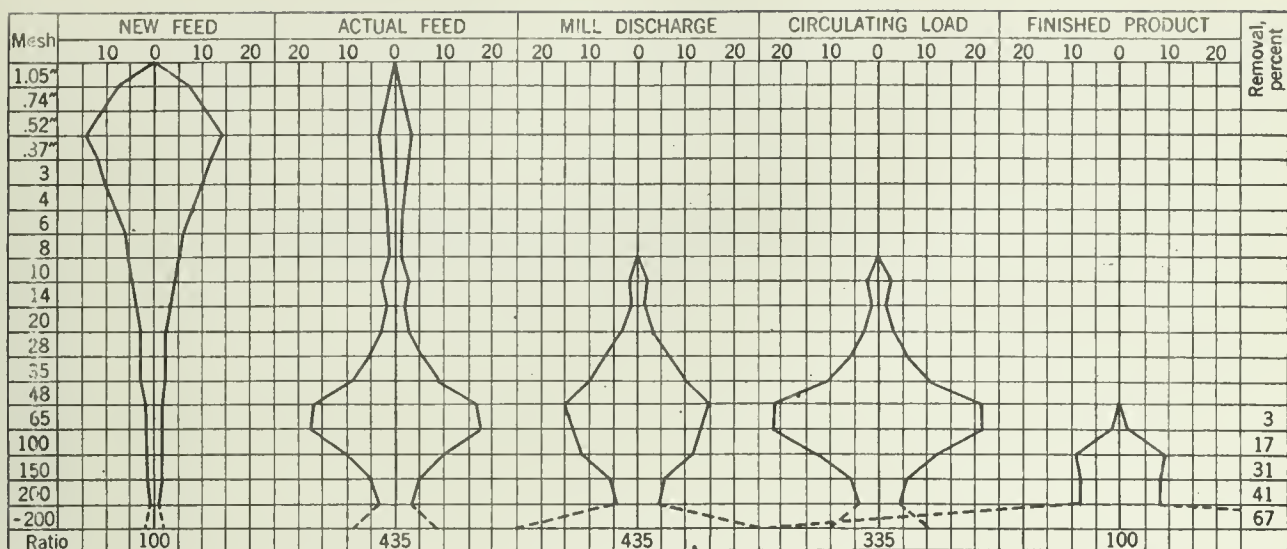


Figure 7.—Sizing diagrams of products of ball mill at Chief Consolidated (21).

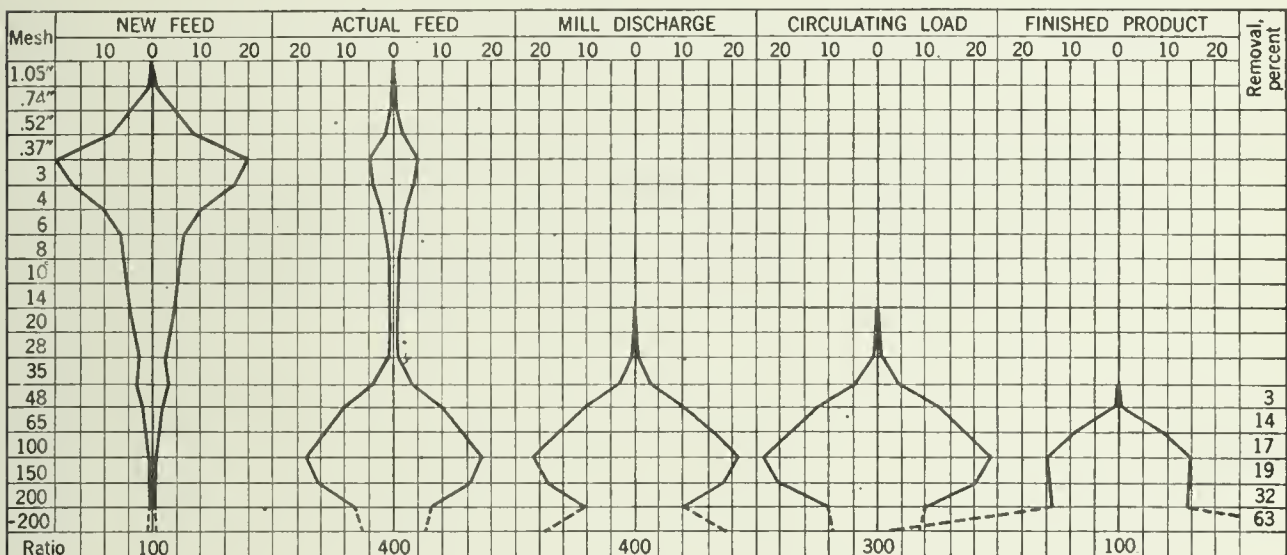


Figure 8.—Sizing diagrams of products of rod mill at Hughesville (27).

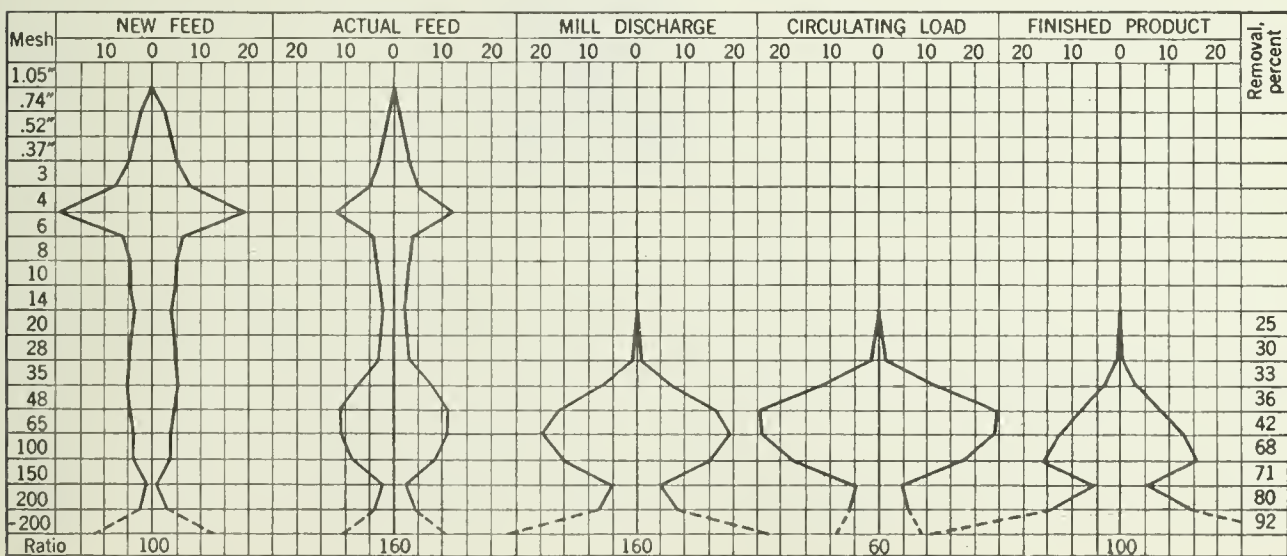


Figure 9.—Sizing diagrams of products of primary rod mill at Midvale (28).

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MILLING METHODS AT THE LEAD-ZINC CONCENTRATOR
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MILLING METHODS AT THE LEAD-ZINC CONCENTRATOR OF THE
INTERNATIONAL SMELTING CO., TOOELE, UTAH ¹

By W. J. McKenna ²

INTRODUCTION

This paper, describing the milling practice at the lead-zinc concentrator of the International Smelting Co., is one of a series of papers on milling methods being prepared by the United States Bureau of Mines.

The concentrator is operated as a unit of the International Smelting Co. and is located about 40 miles southwest of Salt Lake City, at Tooele, Utah. The Tooele Valley Railroad, operating between the plant and Warner, Utah, a distance of 6.5 miles, makes connection with both the Union Pacific and Denver & Rio Grande Railroads.

The concentrator operates as a custom lead-zinc mill, treating approximately 1,150 tons per day entirely by flotation methods. The resulting lead and iron concentrates are smelted locally and the zinc concentrates are shipped to the Anaconda Copper Co. plants at Anaconda and Great Falls, Mont.

In general, the concentrator consists of a frame building housing the coarse crushing, fine grinding, flotation, concentrate thickeners, and filter equipment.

WATER

Fresh water is piped a distance of 5 miles from mountain springs to a 412,000-gallon capacity concrete storage tank.

1 The Bureau of Mines will welcome reprinting of this paper, provided the following footnote acknowledgment is used: "Reprinted from U. S. Bureau of Mines Information Circular 6758."

2 One of the consulting engineers, U. S. Bureau of Mines, and concentrator superintendent, International Smelting Co.

POWER

Electric power is supplied by the Utah Power & Light Co. at 44,000 volts and transformed to 440, 220, and 110 volts for use in the concentrator. Steam power is generated at the centralized smelter power house and distributed to the concentrator at 100 pounds' pressure.

ORE TREATED

The major sources of ore supply have been from the Park City and Bingham, Utah, mining districts. Lesser but important amounts have been received from the Tintic, Frisco, and Ophir, Utah, districts and the adjacent States of Colorado, Idaho, and Nevada. Materials from the above sources include crude sulphide ores and gravity-mill table middlings and slimes.

It is evident that ores shipped from so many different mining districts would vary in general characteristics and metal contents. A typical analysis of a mixture of crude ores and gravity-mill products follows:

Percent				Ounces per ton	
Lead	Copper	Zinc	Iron	Gold	Silver
8.62	0.42	11.23	9.75	0.137	7.10

Owing to the several different ores received, only more or less general descriptions can be given of their mineral contents.

The copper content, while low, is present chiefly as chalcopyrite and tetrahedrite, with minor amounts of covellite and bornite. The lead mineral, galena, varies from the fine-grained steel galena to the coarse crystalline variety, the latter often showing slight surface oxidation. Daily determinations of oxide lead in the mill feed shows an average of 0.10 percent. The precious metals generally follow the copper minerals and galena, although gold is associated with the pyrite to a large extent in most of the ores. Zinc occurs mostly as a true sphalerite, which has a light resin color. The predominating gangue is hard quartzite, although several ores contain soft talc-like gangue which causes considerable difficulty in the settlement of tailing.

Average moisture content of ore receipts is 4.5 percent. Mixing the dry and sticky ores tends to eliminate most of the difficulties experienced in handling them.

HISTORY OF CONCENTRATOR OPERATIONS

Before presenting the historical facts of the concentrator operations, it might be of interest to discuss briefly the problems which confronted the lead-smelting industry of this territory from 1918 until the introduction of selective flotation methods. The oxidized silver ores contained excess quantities of silica and the zinc content of the lead-silver sulphide ores was increasing. The elimination of silica and zinc in lead blast furnace operations required fluxing with iron, but it was difficult to obtain suitable iron ores for this purpose. A careful survey of the situation by the geological department in conjunction with the flotation testing laboratory staff revealed that a selective flotation process would accomplish: (1) The elimination of silica from the lead-silver sulphide ores; (2) the removal of zinc from many ores into a separate concentrate, thus bringing a revenue rather than a penalty to the shipper; (3) the production of an iron concentrate from the pyrite content of the sulphide ores; and (4) the mining of lower-grade ore deposits heretofore considered unprofitable.

With these possibilities in mind, the company pioneered in the establishment of a central custom mill in 1924, purchasing lead-zinc ores on schedules based upon the actual recoveries made and grades of concentrates produced. At this time the company was operating a 1,000-ton-capacity copper flotation concentrator treating the ore from the Utah Delaware mine in two 500-ton-capacity sections. In order to treat approximately 500 tons of lead-zinc iron ores per day it was necessary to use the fine-grinding unit of one 500-ton section and to rearrange the entire Callow equipment into a suitable flow sheet for the operation of three different circuits; namely, lead, zinc, and iron. Installation of additional concentrate thickening and filter equipment was also necessary.

The capacity of the concentrator was increased 500 tons per day early in 1925. Minerals Separation Sub-Aeration machines were selected for the flotation equipment of this addition. The present flow sheet of the mill uses both Callow and Minerals Separation in each section rather than entire sections of each type.

PRESENT CONCENTRATOR PRACTICE

A flow sheet of the crushing plant is given in figure 1; figures 2 and 3 present flow sheets of fine grinding and flotation of section 1, respectively. The treatment in section 2 is similar to that of section 1, except that two 14-cell, 18-inch Minerals Separation machines are used in the zinc rougher circuit in place of the 4-cell M. S. conditioner and four 8-pan Callow cells as shown in figure 3.

HANDLING AND MINING OF ORES

Custom ores are crushed and sampled at the smelter sampling mill. The final product, which is discharged from a Symons cone crusher set at a 3/8-inch opening, is conveyed to one of 10 storage bins having a total capacity of 2,250 tons.

Mixtures are made of the various ores according to the amenability of the individual ores to the general reagent combination as applied to the basic ore being treated at that time, predetermined by batch laboratory tests or previous treatment at the concentrator. The ores entering the mixture are fed from the storage bins onto a long conveyor belt extending the entire length of the bins and discharged into 50-ton-capacity, bottom-dump railroad cars, which are later dumped into a receiving pocket at the concentrator. From the receiving pocket the ore is conveyed and elevated to either section 1 or 2 ball mill feed bins, thereby by-passing the concentrator crushing equipment.

Gravity-mill products received from other mills are first pipe-sampled and then introduced into the regular mill feed, by loading small amounts on the mixture cars by a locomotive crane.

Company account ores shipped from Bingham over the aerial tramway are dumped and crushed at the concentrator. Every fifth bucket, however, is sent to the smelter sampling mill for weighing and sampling.

CRUSHING

Referring to figure 1, apron feeders deliver the ore from the coarse-ore bins at the concentrator to a 30-inch inclined conveyor at the rate of 125 tons per hour. The conveyor is direct driven by a 25-hp. motor and discharges onto a stationary grizzly having 2-inch spaces. The grizzly oversize passes to a no. 6 Tel-smith gyratory crusher with the discharge set at approximately 2 inches. The average life of the manganese-steel mantle and concaves, when handling 575 tons of ore per day, is 2-1/2 years.

The grizzly undersize and crusher product are conveyed to a 30-inch elevator equipped with a double row of 14- by 7-inch manganese-steel buckets, which discharges over two 4- by 6-foot Mitchell Hum-mer screens having 1-1/4-inch holes.

The screen undersize goes to the ball-mill feed bins and the oversize goes to a set of 54- by 20-inch Traylor heavy-duty rolls. The rolls shells are made of forged steel and show a consumption of 0.024 pound per ton of ore crushed. The rolls are operated at a speed of 95 r.p.m. and are belt driven from a line shaft which also drives the crusher. A 150-hp. motor is used for operating the crusher and rolls.

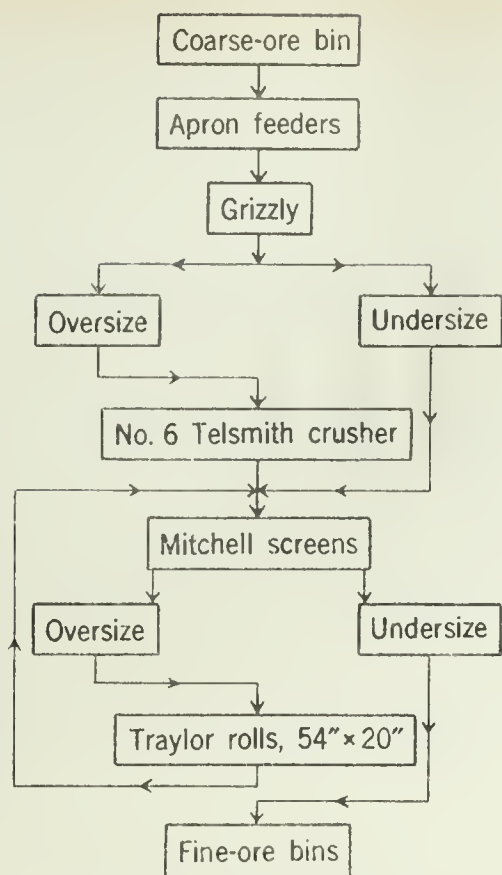


Figure 1.—Flow sheet of crushing plant.

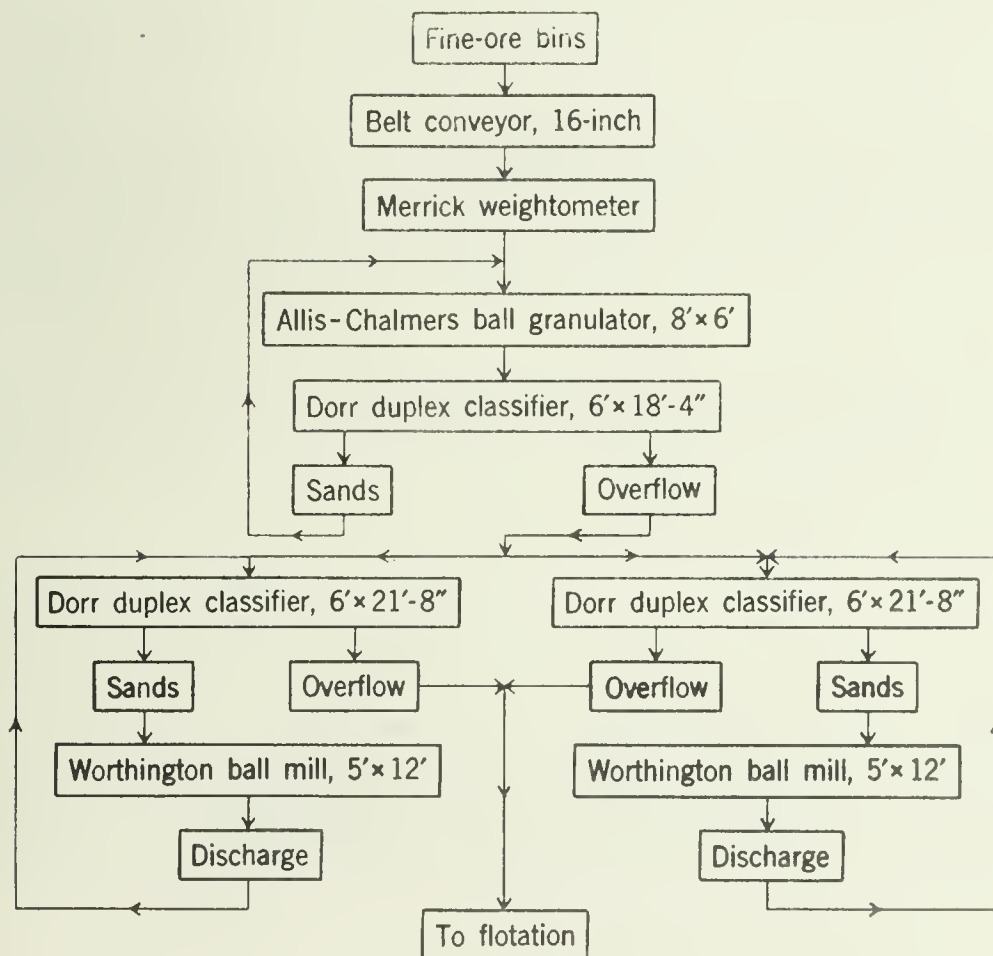


Figure 2.—Flow sheet of grinding plant.

FINE GRINDING

Referring to figure 2, the fine grinding equipment of each 500-ton section is identical and is of the two-stage type. Each section has a 650-ton-capacity fine-ore bin from which feed is drawn by means of ratchet-driven belt feeders discharging onto a 16-inch conveyor belt. The ore is carried by this belt over a Merrick weightometer to the double-scoop feeder of an 8- by 6-foot Allis-Chalmers grate-discharge ball mill, which operates in closed circuit with a 6-foot by 18-foot 4-inch Dorr duplex classifier and constitutes the primary grinding circuit. The overflow from the primary classifier is equally divided between two secondary 6-foot by 21-foot 8-inch Dorr duplex classifiers, each operating in closed circuit with a 5- by 12-foot Worthington ball mill. The overflow from the secondary classifiers comprises the flotation feed.

The primary mills are operated at a speed of 18.7 r.p.m. and are driven by 200-hp. induction motors, connected by means of Link-Belt drives. Cast-iron balls, 3-inch diameter, are used and are manufactured at the foundry of the Anaconda Copper Mining Co. at Anaconda, Mont. Ball loads amount to 27,000 pounds. Feed end and shell liners are made of manganese steel and the grate sections are made of tool steel. Pulp density in the primary mills is maintained between 72 and 75 percent of solids.

The secondary mills operate at a speed of 27.2 r.p.m. and are driven by 100-hp. induction motors, as well as through Link-Belt silent-chain drives. Balls used are 2-inch cast iron, also manufactured at Anaconda, Mont. Ball loads carried are 14,000 pounds per mill. Feed end and shell liners are made of hard white iron. Pulp density in the secondary mills is maintained between 68 and 70 percent of solids.

The following tabulation shows liner and ball consumptions, in pounds per ton of primary mill feed, for the primary and secondary mills:

	Liners and grates	Balls
Primary mills, pounds	0.30	2.30
Secondary mills, pounds20	1.14

Screen analyses of grinding-circuit products are given in table 1. It should be noted that the figures given in table 1 represent a period in which extra fine grinding was practiced. More typical examples of screen sizes of classifier overflow and flotation tailing are presented in table 2.

Table 1. - Screen analyses of grinding-circuit products

Screen size, mesh	Weight, percent						
	Primary ball mill feet	Primary classifier sands	Primary ball mill discharge	Primary classifier overflow	Secondary classifier sands	Secondary ball mill discharge	Secondary classifier overflow
+ 48	75.3	61.0	30.2	12.7	14.4	1.5	--
+100	6.0	23.7	19.7	20.4	33.4	8.7	3.9
+200	4.8	7.6	13.9	15.7	28.3	22.6	12.6
-200	14.9	7.7	36.2	51.2	23.9	67.2	83.5

Table 2. - Typical screen analyses

Screen size, mesh	Weight, percent	
	Secondary classifier overflow	Flotation tailing
+ 65	1.0	2.0
+100	5.6	9.0
+150	7.4	12.0
+200	11.6	14.6
-200	74.4	62.4

CLASSIFICATION

The 6-foot by 18-foot 4-inch primary classifiers have a slope of 3-1/2 inches per foot and operate at a rate of 26 strokes per minute. The 6-foot by 21-foot 8-inch secondary classifiers slope 2-3/4 inches per foot and their rakes operate at a speed of 22 strokes per minute. The secondary classifier overflow pulp which is flotation feed is maintained between 22 and 24 percent of solids.

FLOTATION

Referring to figure 3, the secondary classifier overflow pulp from each grinding section is conveyed by launder directly to flotation. Present practice follows the use of two 14-cell 18-inch Minerals Separation Sub-Aeration machines for the lead circuit roughers of each section. The zinc rougher circuit of section 1, as previously described, consists of four 3-pan Callow cells, while the zinc rougher circuit of section 2 employs two 14-cell 18-inch Minerals Separation Sub-A. machines. Callow cells are used in the iron rougher circuits of both sections and also for cleaning all lead, zinc, and iron rougher concentrates produced in either section.

The line shaft of each M. S. machine is driven by a 40-hp., silent chain drive at a speed of 380 r.p.m. The spindle shafts are driven at a speed of 331.3 r.p.m. from the line shaft through bevel gears. The peripheral speed of impellers is 1,560 feet per minute.

Impellers are 18 inches in diameter, full double-disk type and are made of hard white iron. Their average life is 5 to 7 months, depending upon whether in lead- or zinc-circuit service.

Callow cell pans are the standard 3- by 3-foot type equipped with 3-ply 18-ounce canvas blankets. Average life of cell blankets is 90 days.

Rougher lead concentrate is removed from the first four to six cells of the M. S. machines and cleaned in a 3-pan Callow cell. The cleaner tailing, together with the froth from the remaining cells of the M. S. rougher machines, is returned by bucket elevator to the head of the lead circuit. Tailing pulp from the lead circuit is pumped to a 30-foot Dorr thickener from which the underflow is conveyed by gravity to the head of the zinc-rougher circuit. The Dorr thickener functions as a pulp conditioner, giving the pulp a longer time of contact with flotation reagents, and also as a regulator of the density of the feed to the zinc circuit.

Steam is introduced into the lead-circuit tailing pulp before it enters the zinc-rougher cells. The temperature of the pulp in the zinc circuit is maintained at about 30° C.

In section 1, a conditioner is provided between the Dorr thickener and the Callow zinc circuit. The conditioner is a 4-cell, standard M. S. machine, with 24-inch cruciform agitators driven by a 25-hp. motor through Tex-ropes. The flow sheet of section 2, which employs M. S. Sub-A. machines in the zinc circuit, contains no conditioner between the Dorr thickener and the zinc circuit.

The froths from the first 3 pans of each Callow zinc rougher are combined and double-cleaned in 6-pan Callow cells. The froths from the first four to six cells of each of the section 2 zinc rougher machines are combined and double-cleaned in two 3-pan Callow cells.

Middling and cleaner tailing from the zinc circuits of each section join the tailing from the rougher lead circuits of their respective sections and are pumped to the 30-foot Dorr thickeners previously mentioned.

Tailing from the zinc-rougher circuit of each section comprises the feed to the iron circuit. In section 1, rougher-iron concentrates are taken from the first four pans of each of two parallel 12-pan Callow cells. The rougher concentrate is cleaned in a 6-pan Callow cell. Rougher-iron concentrates in section 2 are taken from the first 3 pans of each of two parallel 8-pan Callow cells and cleaned in a 6-pan cell. Middling and cleaner tailing are returned to the head of the iron circuit of each section. Tailing from the iron circuits of each section is thickened and sent to the tailing disposal pond.

REAGENTS

Soda ash is used in amounts of from 0.1 to 2.0 pounds per ton of ore as a conditioning reagent. It is added to the primary ball mill and to the lead-circuit tailing.

Sodium cyanide is fed to the primary ball mill as a 20 percent solution, in amounts of from 0.2 to 0.3 pound per ton of ore.

Zinc sulphate is fed to the primary ball mill as a 15 percent solution in amounts varying from 0.6 to 1.3 pounds per ton of ore.

Sodium aerofloat is fed dry to the primary ball mill in amounts of 0.1 pound or less per ton of ore.

Potassium ethyl xanthate is fed as a 15 percent solution to the heads of the lead and zinc circuits in amounts of 0.03 to 0.04 pound per ton of ore and to the head of the iron circuit in amounts of 0.15 to 0.30 pound per ton of ore.

Aerofloat No. 15 is used at an average rate of 0.15 pound per ton of ore as a collector and frother in the zinc circuits.

Copper sulphate is added dry, in fine crystalline form, to the lead tailing ahead of the zinc circuit at the rate of 1.25 to 1.50 pounds per ton.

Potassium amyl xanthate is fed to the iron circuit as a 15 percent solution.

All flotation reagents which are fed in aqueous solution are distributed from disk-and-cup feeders, there being a separate feeder for each point of addition. Aerofloat is fed from either disk-and-cup or Geary feeders. Dry reagents, such as copper sulphate, soda ash, and sodium aerofloat are fed from belt-and-hopper feeders.

BLOWERS

Three No. 6-1/2 Roots blowers, two of which are operated at a time, furnish air for the Callow and Minerals Separation Sub-Aeration flotation equipment and the Oliver filters.

These blowers each have a capacity of 5,200 cubic feet of air per minute at 4-1/2 pounds' pressure. The air is distributed to the Callow cells and Oliver filters at a pressure of 4-1/2 pounds per square inch and to the M. S. machines at 16 ounces per square inch. Average volumes of air used are 7 cubic feet per minute per square foot of Callow cell pan area and 20 cubic feet per minute per cell for the M. S. machines.

The blowers are located in the grinding division of the concentrator and are all connected to a main header pipe from which air is distributed to the flotation and filtering divisions. Each blower is belt driven by a Lenix drive at a speed of 225 r.p.m. by a 150-hp. induction motor.

METALLURGICAL RESULTS

Average analyses of heads and products of flotation operations and average recoveries are given in table 3. Metallurgical data for February 1931 are presented in table 4.

DEWATERING AND HANDLING CONCENTRATES

The equipment for concentrate thickening includes three 30- by 10-foot and one 40- by 10-foot Dorr thickeners. One of the 30- by 10-foot thickeners handles lead concentrates from both sections, another of the same size is used for the iron concentrate, and the 40- by 10-foot thickener is used for the zinc concentrates of both sections. A spare 30- by 10-foot thickener is available for emergency use or for the separate thickening of any product from either section if desired.

The pulp density of the concentrate pulps entering their respective thickeners is approximately 20 percent of solids. Thickened pulps containing 50 to 60 percent of solids are elevated to Oliver drum-type filters. The thickener overflows are combined and form an important portion of the return circuit water.

The filtering equipment consists of four 12- by 10-foot and one 12- by 12-foot Oliver drum-type filters. It is seldom necessary to use more than one of these filters for each concentrate. However, the two extra filters permit the filtering of special concentrates and eliminate lost time in operation when repairs are necessary.

It is customary to thicken and filter a portion of the iron concentrate with the lead concentrate, the amount depending upon that desired for sintering and fluxing operations at the smelter.

The moisture content of the filtered concentrates follows:

	<u>Percent moisture</u>
Zinc concentrate	5.3 to 10.0
Lead-iron concentrate	6.5 to 9.5
Iron concentrate	7.5 to 10.0

The variation in moisture content is due to the character of the gangue in the ore treated. The filters are operated at a speed of one revolution in 10 minutes and have an average capacity of 800 pounds per square foot per 24 hours.

Table 3. - Average analyses of flotation products and mill recoveries

	Analyses							Recovery or distribution, percent						
	Percent						Ounces per ton							
	Weight, percent	Lead	Copper	Zinc	Iron	Insol- uble	Gold	Silver	Lead	Copper	Zinc	Iron	Gold	Silver
Heads	100.00	8.62	0.42	11.23	9.75	---	0.137	7.10	100.00	100.0	---	---	100.00	100.00
Lead concentrate	12.60	64.02	2.00	7.67	3.83	3.33	.539	44.65	93.62	60.8	8.61	4.96	49.71	79.26
Zinc concentrate	16.41	1.19	.65	58.44	3.28	2.50	.116	3.67	2.26	25.8	85.44	5.52	13.86	8.49
Iron concentrate.	15.74	.80	.18	2.71	41.50	5.37	.188	2.85	1.47	6.7	3.80	67.00	21.64	6.31
Final tailing ..	55.25	.41	.05	.44	3.98	---	.037	.76	2.65	6.7	2.15	22.52	14.79	5.94

Table 4. - Metallurgical data for February 1931

Total ore treated	tons	10,434.20
Days operated		23
Hours operated per day		19.26
Average ore treated per 24 hours operating time	do.	565.29
Average number of sections operating		1
Total lead concentrate produced	do.	1,315.18
Average lead concentrate produced per 24 hours	do.	71.25
Total zinc concentrate produced	do.	1,712.62
Average zinc concentrate produced per 24 hours	do.	92.78
Total iron concentrate produced	do.	1,642.11
Average iron concentrate produced per 24 hours	do.	88.96
Total concentrates produced	do.	4,669.91
Average concentrates produced per 24 hours	do.	252.99
Ratios of concentration, tons into 1:		
Lead circuit		7.93
Zinc circuit		6.09
Iron circuit		6.35
Combined circuits		2.23
Consumption of reagents per ton of ore treated		
Soda ash	pounds	1.85
Sodium aerofloat065
Sodium cyanide217
Zinc sulphate738
Copper sulphate		1.379
Aerofloat No. 15100
Potassium ethyl xanthate314
Potassium amyl xanthate115

A vacuum of 21 inches of mercury is maintained by two 22- by 8-inch Ingersoll-Rand type ER-1 vacuum pumps. Air for blowing is furnished by the flotation blowers at 4-1/2 pounds' pressure per square inch.

Filter covers are of Palma twill canvas cloth and have an average life of 75 days.

The filter cake is discharged directly into railroad cars. Filtrate water is reclaimed and used in the return circuit water.

DISPOSAL OF TAILINGS

The final tailings from both sections, amounting to about 50 percent of the weight of ore treated, are conveyed by launder to a 100-foot diameter Dorr thickener and thickened from 15 to 50 percent of solids. The overflow water amounting to approximately 350 gallons per minute is added to the return water circuit. The thickened tailing pulps are elevated by a 4-inch Wilfley pump to a launder and flow by gravity to impounding ponds located about 1-1/2 miles below the concentrator. No attempt is made to reclaim any water from these ponds.

SAMPLING

During each shift samples are taken of heads, lead concentrate, zinc concentrate, iron concentrate, and tailing of each section with standard automatic samplers. These samples are assayed daily for the purpose of metallurgical control and guidance.

WATER SUPPLY

Fresh water, as previously stated, is obtained from mountain springs and has the following analysis:

	<u>Grains per U. S. gallon</u>
Organic matter	7.47
Fe ₂ O ₃ and Al ₂ O ₃35
SiO ₂58
CaO	6.88
MgO	3.73
SO ₃	5.13
Total solids	32.67

The consumption of fresh water for metallurgical purposes amounts to approximately 115 gallons per minute or is at a rate of 144 gallons per minute with a feed rate of 1 ton of ore per minute. This amount corresponds closely to the loss of water in the thickened tailing pulp and as moisture in the concentrates. The introduction of fresh water is chiefly as spray water in the rougher lead and zinc concentrate launders.

All possible water is reclaimed from thickening and filtering operations and concentrated at a convenient elevation, from which it is pumped to two equalizing tanks located above the mill at an elevation sufficient to permit its use in any division.

Two 6-inch Cameron pumps are used for circulating the return water, each of which is direct-driven by a 50-hp., 1,735 r.p.m. motor.

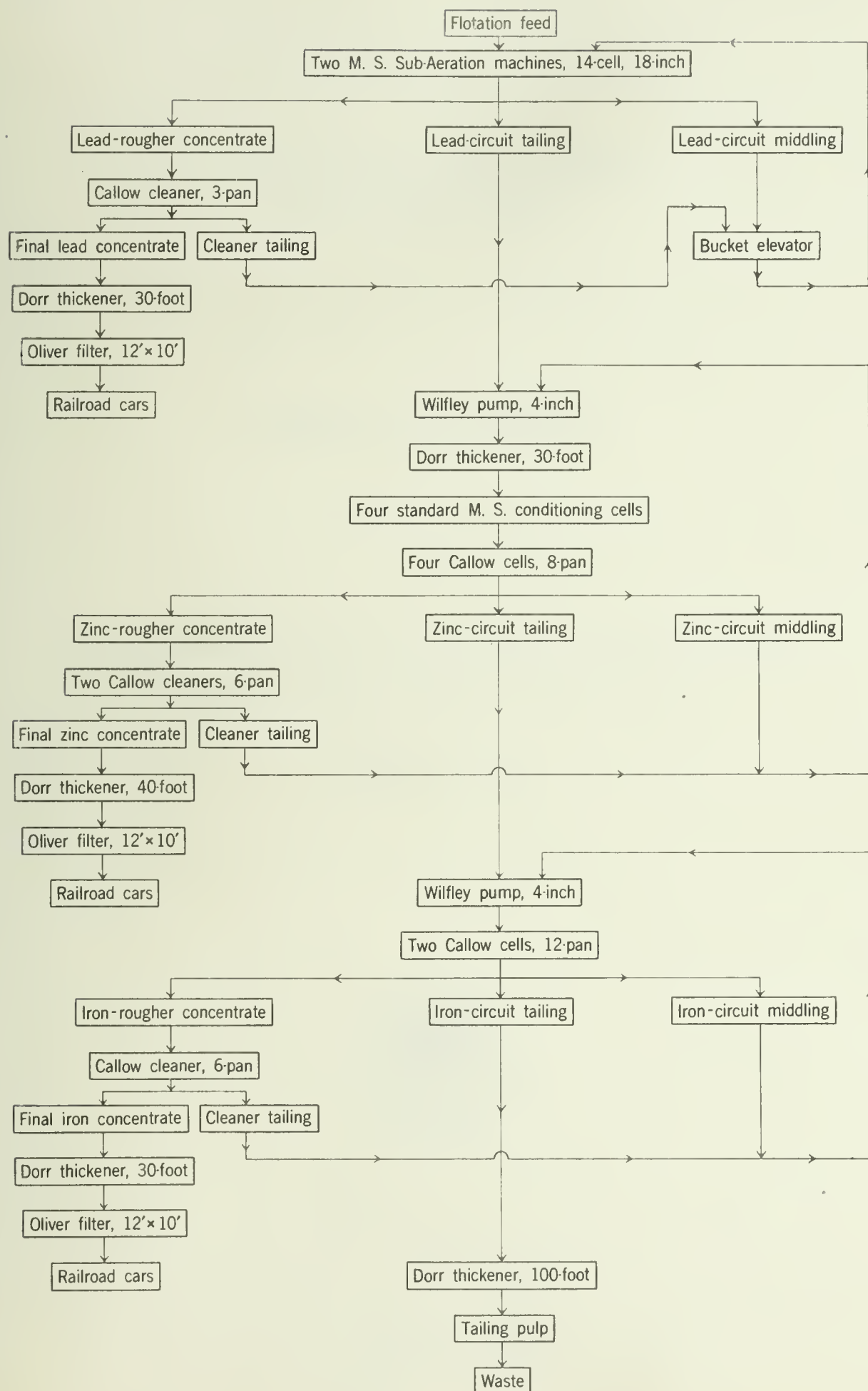


Figure 3.—Flow sheet of flotation and dewatering of section 1.

UNITED STATES BUREAU OF MINES
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INFORMATION CIRCULAR

MILLING METHODS AT THE OXIDE CONCENTRATOR
OF THE INTERNATIONAL SMELTING CO.,
TOOELE, UTAH



BY

J. J. BEAN

I.C. 6759
December, 1933.

INFORMATION CIRCULAR
UNITED STATES BUREAU OF MINES

MILLING METHODS AT THE OXIDE CONCENTRATOR OF THE
INTERNATIONAL SMELTING CO., TOOELE, UTAH^{1/}

By J. J. Bean^{2/}

INTRODUCTION

This paper, describing the milling practice at the oxide concentrator of the International Smelting Co., is one of a series on milling methods and costs being published by the United States Bureau of Mines.

GENERAL DESCRIPTION

The oxide concentrator is properly an integral part of the Tooele plant of the International Smelting Co., located 5 miles from Tooele, Tooele County, Utah.

It has a capacity of 300 tons of ore per day and uses an all-flotation process to treat custom siliceous oxidized lead ores obtained principally from the Tintic district in Utah. Ores found amenable to the process are purchased on a custom basis, the principal tonnages being obtained from the Tintic Standard and North Lily mines.

The concentrator is located on a hillside. It comprises fine-grinding, flotation, filtering and drying equipment, together with the necessary conveying, elevating, and pumping equipment necessary to connect all these departments and to supply crude ore to the fine-ore bins. No crushing or dry-sampling equipment is provided, as all ore is crushed by a Symons cone crusher set at three-eighths inch before going to the concentrator, either by a custom sampling works or by the smelter sampling mill.

Water is supplied from a reservoir previously constructed to serve the lead-zinc concentrator at the plant. This reservoir is supplied with water by various springs in canyons adjacent to the plant.

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- ^{1/} The Bureau of Mines will welcome reprinting of this paper, provided the following footnote acknowledgment is used: "Reprinted from U. S. Bureau of Mines Information Circular 6759."
- ^{2/} One of the consulting engineers, U. S. Bureau of Mines, and assistant concentration engineer, International Smelting Co.

Electric power is used for operation and is supplied by the Utah Power & Light Co. at 44,000 volts. The voltage is reduced to 2,200 by an outside sub-station, at which voltage all motors above 50 hp. are supplied. For smaller motors and lighting the voltage is further reduced to 440 or 110 by suitable transformers within the mill itself.

ORES TREATED

As previously stated, the ores treated are from the Tintic Standard and North Lily mines. They may be characterized as oxidized, highly siliceous lead-silver-bearing ores of an earthy brown color. The following is a typical analysis:

Copper	percent...	0.21
Lead	do. ...	18.95
Zinc	do.78
Iron	do. ...	6.50
Lime	do. ...	4.17
Insoluble	do. ...	56.80
Gold	ounces per ton...	.05
Silver	do. ...	23.38

The copper occurs chiefly as chalcopyrite, chalcocite, and tetrahedrite. The lead occurs as galena, cerussite, anglesite, and to a lesser extent plumbojarosite, the first two minerals mentioned predominating. The principal silver mineral is cerargyrite; appreciable amounts of proustite and argento-jarosite are also present. The gold is associated with the principal copper and lead minerals and pyrite; varying amounts are present as tarnished native gold. The gangue is principally a rather opaque quartzite but also includes oxidized iron minerals, dark colored cherts, lime, and gypsum. The ore is readily ground but owing to the high cerussite content is difficult to classify efficiently.

HISTORY OF CONCENTRATOR OPERATIONS

Present flotation operations are the outgrowth of an unbalanced silica load at the lead smelter and a resulting desire to reject some of the silica in the oxidized siliceous ores received at the smelter before sending them to the lead plant for smelting.

Various methods of concentrating were tested at the company concentration laboratory before the present successful method was evolved in the fall of 1927. During the winter of 1927-28 various ores that might be available were tested in the concentration-department continuous testing plant, and in March 1928, a full-scale mill test was commenced, using a section of the lead-zinc concentrator. After several months of this operation, during which time many operating difficulties were presented and overcome, sufficient information was obtained to design a mill especially for the treatment of oxidized ores.

Construction of the present mill was commenced in February 1929, and the mill started operation in the latter part of June of the same year. Operation was continuous until the end of 1929, when depressed metal markets resulted in intermittent operations until the present time. Operations in the new concentrator have been successful from the first, and only reduced tonnages of ore have caused the irregular operation.

PRESENT METHOD OF CONCENTRATING

Referring to figure 1, ore as received either from the custom sampler or the smelter sampling mill is dumped into any or all of twelve 125-ton-capacity receiving bins arranged in two parallel rows of six bins each. From these bins it may be drawn out in any desired proportion onto a collecting conveyor by means of constant-speed feeder belts on the bottoms of the bins and variable opening gates on the bins themselves. This collecting conveyor belt discharges onto a cross belt which conveys the ore to the boot of a 30-inch elevator equipped with a double row of 14- by 7-inch malleable iron buckets. This elevator discharges the ore upon a 20-inch inclined belt which conveys it 504 feet over the tracks of the Tooele Valley Railway and up to the ball-mill feed bins at the top of the mill.

There are two ball-mill feed bins constructed of steel and having a capacity of 200 tons each. The No. 1 bin is filled directly from the inclined conveyor, while the No. 2 bin is filled from the inclined conveyor by means of a short horizontal conveyor belt 20 inches wide, running across the tops of the bins and discharging at the center of the No. 2 bin. Both feed bins are equipped with feeder belts and gates so that any convenient rate of discharge may be obtained. Both feeder belts discharge onto the ball-mill feed belt which is on a slight incline and discharges into the ball-mill scoop housing. Ore on this feed belt is weighed on a Biehler weightometer before passing to the ball mill.

GRINDING

Fine grinding is done in one No. 86 Marcy ball mill operating in closed circuit with an 8-foot by 23-foot 4-inch Dorr duplex classifier, model D. Normal feed rate is 300 wet tons per 24 hours. Cast-steel balls, 3-inch diameter, manufactured by the Anaconda Copper Co. at its Anaconda foundry, are used. Inasmuch as the ball mill has considerable capacity in excess of the flotation equipment, a somewhat light ball load of about 11 tons is carried. The pulp in the ball mill is carried at a density of about 80 percent of solids; that of the classifier overflow is maintained at about 26 percent of solids.

Screen analyses of ball-mill feed, classifier overflow, and flotation tailings are given in table 1.

FLOTATION

Flotation is carried on in four 18-inch M.S. Sub-A flotation machines, operating in parallel, the feed being provided to the four machines by means of a mechanical distributor of a design developed by the company and built in its shops. Each machine has 14 cells, the first three of which are used for flotation of sulphide minerals and the last 11 of which are devoted to the flotation of oxidized minerals.

Table 1. - Screen analyses of concentrator products

Screen size	Weight, percent		
	Ball-mill feed	Classifier overflow	Flotation Tailings
+0.525 inch	0.64	--	--
+0.371 do.	4.89	--	--
+ 3 mesh	9.57	--	--
+ 4 do.	11.75	--	--
+ 6 do.	9.98	--	--
+ 8 do.	8.96	--	--
+ 10 do.	4.48	--	--
+ 14 do.	.14	--	--
+ 20 do.	.08	--	--
+ 28 do.	15.67	--	--
+ 35 do.	3.88	--	--
+ 48 do.	3.49	--	--
+ 65 do.	3.04	1.6	2.8
+ 100 do.	3.84	7.2	13.0
+ 150 do.	2.53	8.2	14.4
+ 200 do.	2.96	10.2	15.0
- 200 do.	14.10	72.8	54.8

Referring to figure 2, the feed to a flotation machine enters the second cell through a feed box and pipe; ethyl xanthate, amyl xanthate, and sodium silicate are also fed to the feed box. The second and third cells are sulphide roughers, the concentrates from these two cells being cleaned in the first cell. Cleaner cell tailings flow, as usual, to the second cell. The fifth and sixth cells produce finished oxide concentrates; cell 7 to 9, inclusive, produce rougher concentrates which are cleaned in the fourth cell. The froths of the remaining cells, 10 to 14 inclusive, are returned to the head of the oxide circuit for retreatment. Low-head air lifts are used for the return of all middlings and concentrates which require pumping in order to get them to their proper destinations.

REAGENTS

The reagents used comprise sodium amyl xanthate, 1.5 pounds per ton; sodium sulphide, 6.0 pounds per ton; sodium silicate, 4.0 pounds per ton; ethyl xanthate, 1.2 pounds per ton; and pentasol, used as a frother, 0.30

pound per ton. The stage addition of reagents into the early part of the flotation operation is practiced, the principal amounts being added to the fifth cell which is the head of the oxide circuit.

All water-soluble reagents are fed in solution, being dissolved in large tanks on the top floor of the mill. From these mixing tanks they are drawn to the proper feeders, which are located on a floor above the flotation machines. The feeders used were developed at the plant and are of the disk-and-cup design. By means of extra long pins in the disk they feed to six places at one time. Inasmuch as the amyl xanthate and sodium sulphide must be added in stages to effect efficient flotation, enough feeders for these reagents are provided to feed to each cell down to the twelfth cell, if necessary. Other reagents require less complete staging and are consequently provided with fewer feeders.

Changes in operating conditions are met by varying the amounts of sodium sulphide added to the various cells, the amounts of ethyl xanthate, amyl xanthate, sodium silicate, and pentasol remaining essentially constant over a wide range of different ores.

AIR

Air is supplied to the flotation machines at a pressure of 1 pound per square inch by either of two No. 5 Roots blowers. Either blower has greater capacity than is needed, so the other one is held in reserve. High-pressure air for the air lifts is supplied from the smelter high-pressure service line.

METALLURGICAL RESULTS

Analyses and percent recovery or distribution in flotation heads, concentrate, and tailing for September 1930, are given in table 2.

Table 2. - Analyses and recovery or distribution in concentrator products

	Weight, percent	Analyses						Recovery or distribution, percent					
		Percent				Ounces per ton		Copper	Lead	Iron	Insoluble	Gold	Silver
		Copper	Lead	Iron	Insoluble	Gold	Silver						
Heads	100.0	0.18	18.43	7.32	56.35	0.0567	24.85	--	--	--	--	--	--
Lead concen- trate	37.03	.36	47.02	13.94	16.73	.114	58.76	72.2	94.6	43.7	11.0	74.6	87.7
Tailing	62.91	.08	1.53	6.56	79.59	.023	4.87	27.8	5.4	56.3	89.0	25.4	12.3

DEWATERING AND HANDLING CONCENTRATES

The finished concentrates are conveyed to the boot of an elevator which elevates them high enough to allow them to run by gravity to a 50-foot Dorr tank adjacent to the filter building for thickening before filtering. A 3-inch Wilfley pump is also available for this service and is used as a stand-by for use in emergencies.

The concentrate pulp, thickened from about 25 to 50 percent of solids, is discharged directly to the boot of an elevator which conveys it to the top of the filter building where it flows by gravity to any or all of the three 6-leaf 8-foot diameter American filters. A 3-inch Wilfley pump is provided to replace the elevator in case of emergency. The filters discharge onto a reversible conveyor belt which in turn discharges either to a dryer or onto a car-loading belt which also serves the dryer.

A vacuum of 20 to 23 inches of mercury is maintained at the filters by one of two 20- by 12-inch duplex Ingersoll-Rand vacuum pumps. The filters have a capacity of 235 tons per filter day of 24 hours; moisture in filter cake varies from 13 to 15 percent.

In case of failure at the dryer or if the concentrates are to be stored in stockpiles, the reversible conveyor under the filters is set to discharge directly onto the car-loading conveyor and the filter cake, without further drying, is thus loaded directly to railroad cars.

Because of the sticky character of the concentrates after filtering drying is necessary to produce a product that may be handled at the lead-plant bins. For this purpose an 8- by 72-foot Vulcan dryer of the rotary-kiln type is provided. This dryer is brick-lined and gas-fired, natural gas being used as fuel. Filter cake from the filters on the floor above is fed into the upper end of the dryer through a chute, and as it works down the dryer to the discharge end is dried by a current of hot gas traveling countercurrent from the gas-fired furnace at the discharge end. Moisture in final product is held at about 6.5 percent in the interest of economy. With such a final moisture a granular product is obtained which is easily handled in the bins at the smelter.

DUST COLLECTOR

A cyclone-type dust collector is provided in the dryer stack to prevent excessive dusting losses in flue gases. The dust recovered per day amounts to about 300 pounds.

The dryer as previously mentioned discharges onto the car-loading conveyor which carries the concentrate to railroad cars for transportation to the smelter.

DEWATERING AND DISPOSAL OF TAILINGS

Tailings flow by gravity from the flotation machines to a 50-foot Dorr thickener placed in line with and about 100 feet from the thickener building. The pulps are thickened only enough to supply sufficient water for milling, which means a spigot product containing about 40 percent of solids. This sounds like a peculiar statement, but due to the fact that considerable fresh water in the form of compressor, vacuum-pump, and reagent-mixer cooling water enters the circuit at all times the return water is in the nature of "make-up" water.

The overflow from the thickener, together with concentrate-tank overflow, is pumped to an equalizer tank above and in the rear of the concentrator by two 5-inch, type LV, Cameron single-stage pumps direct driven by a 50-hp., 2,200-volt, 1,750-r.p.m. motors. Only one pump is required for this service at one time.

The spigot product of the thickener is discharged directly into the tailing launder and flows by gravity to the tailing pond. No attempt is made to reclaim water from this pond.

SAMPLING

All sampling is done by Galligher automatic samplers. The heads are sampled just after overflowing the classifier, the concentrates just after leaving the flotation machines, and the tailing just after leaving the flotation machines. The products are sampled as a whole, no attempt being made to take regular samples of the products of the individual flotation machines. The products of each shift are sampled and assayed separately so that each foreman knows the metallurgical results which were obtained on his shift, and thus can better control daily operation.

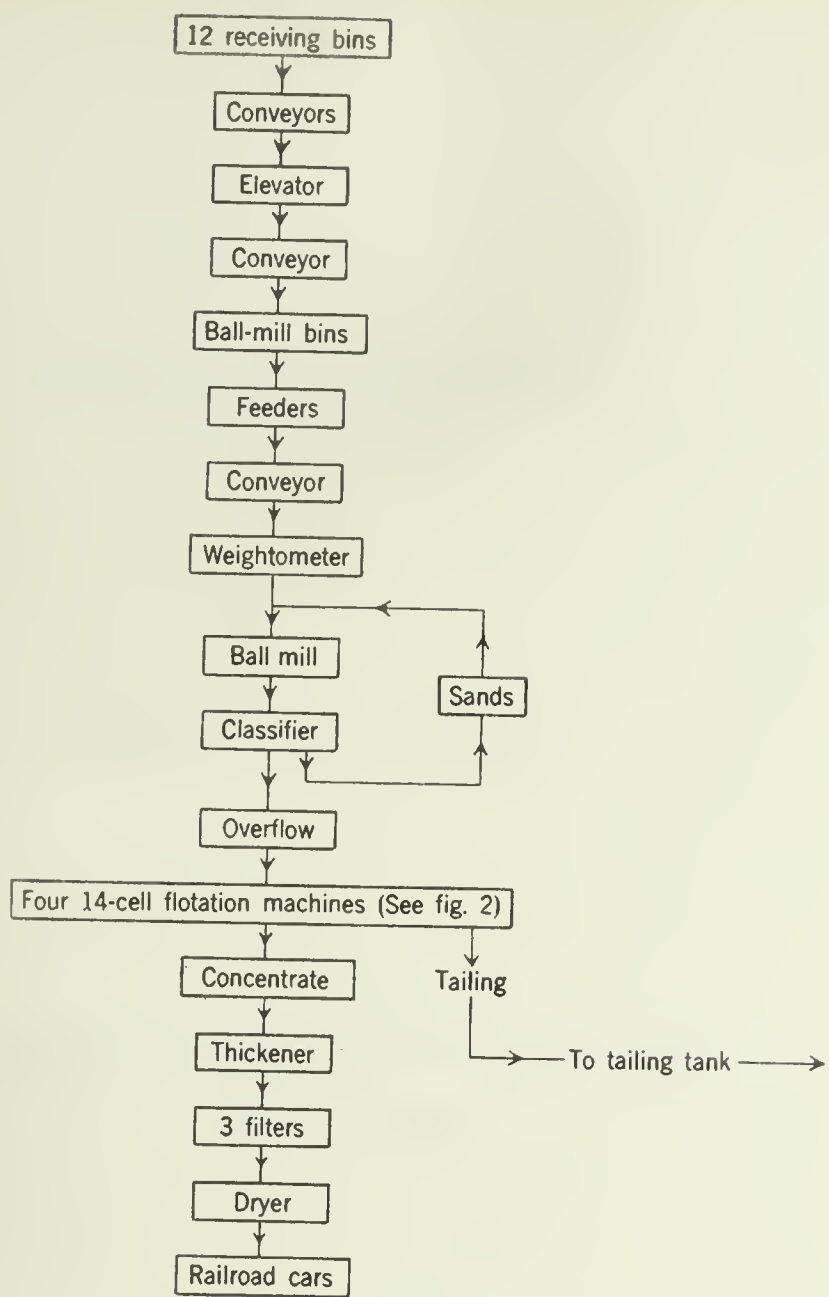


Figure 1.—Flow sheet of ore treatment.

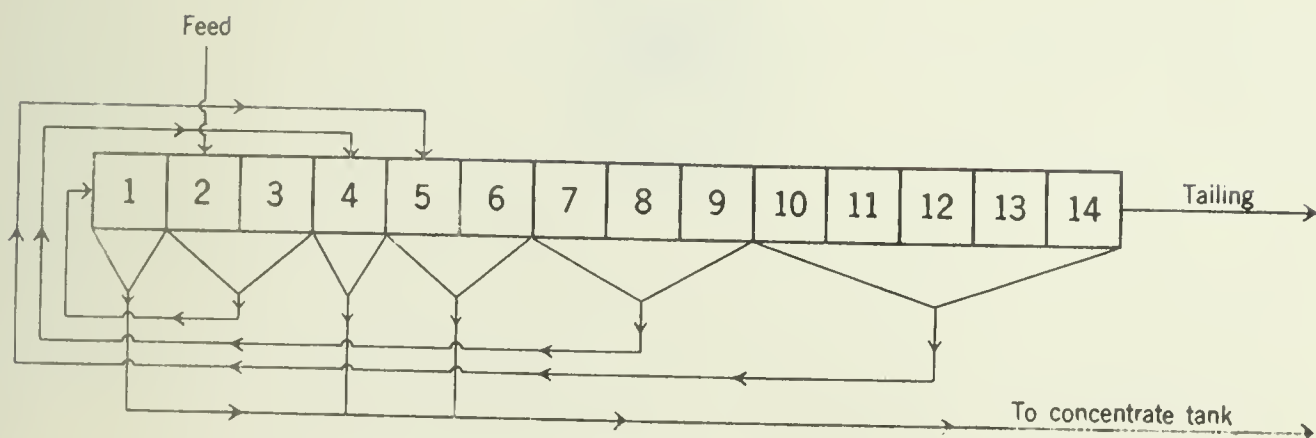


Figure 2.—Method of operating flotation machines.



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FEBRUARY, 1934

UNITED STATES BUREAU OF MINES
SCOTT TURNER, DIRECTOR

INFORMATION CIRCULAR

EXPLOSIONS IN NEW MEXICO COAL MINES,
1895 TO 1932



BY

G. M. KINTZ

INFORMATION CIRCULAR

UNITED STATES BUREAU OF MINES

EXPLOSIONS IN NEW MEXICO COAL MINES
1895 TO 1932 1/

By G. M. Kintz 2/

INTRODUCTION

From 1895 to 1932, inclusive, 1,016^{3/} men were killed in and about New Mexico coal mines. Of these, 473 (approximately 47 percent) were killed in the 17 explosions listed in this paper. The second largest explosion of record in the United States, in so far as fatalities are concerned, occurred in New Mexico.

The data in this circular were taken from publications of the United States Bureau of Mines and published annual reports of the State inspector of coal mines of New Mexico. The tables included are complete only with regard to major disasters. The Bureau of Mines considers as a major disaster one in which 5 or more men are killed. A few minor disasters are included, however.

This is one of a series of papers published by the Bureau of Mines to give by States the data available regarding coal-mine explosions in the hope that the information may assist in eliminating similar conditions of potential danger now existing in many coal mines of the United States.

EXPLOSIONS

Tables 1 and 2 give available data on the cause, number of men killed and injured, and possible means of prevention at the time of 17 explosions in New Mexico from 1895 to 1932, inclusive.

Nature of explosions.— Seven of the explosions listed occurred before 1913. The nature of the explosions with reference to gas and coal dust is stated in table 3.

1/ The Bureau of Mines will welcome reprinting of this paper, provided the following footnote acknowledgment is used: "Reprinted from U. S. Bureau of Mines Information Circular 6760."

2/ Associate mining engineer, U. S. Bureau of Mines Safety Station, Denver, Colo.

3/ This figure is compiled from U. S. Bureau of Mines Bulletins 115, 275, and 355 and C.M.F. 18.

In the 4 gas explosions listed 15 men were killed and 9 injured; in the 7 dust explosions 400 men were killed and only 6 injured; in the 4 gas and dust explosions 34 men were killed and 9 injured.

These figures illustrate that usually where coal dust is a factor in an explosion the destruction of life and property is much greater than when gas alone is the exploding medium. They show the necessity of thorough rock-dusting, with sprinkling of working places and of haulage trips where possible, to prevent participation of coal dust in explosions in bituminous and lignitic coal mines. A gas and dust explosion is likely to be more disastrous than either a gas or dust explosion alone; in fact, nearly all widespread coal-mine explosions have involved both gas and dust. Some coal mined in New Mexico has dust which rates among the most inflammable of the coal dusts of the United States. In addition many, if not most, of these mines also give off explosive gas of a decidedly dangerous combination, which is reflected in the very bad record of New Mexico as to fatalities from coal-mine explosions. The need for strict enforcement of the rock-dusting clause in the mining code recently enacted by the New Mexico legislature is also indicated by these facts and figures.

Explosives used.— The use of black blasting powder or dynamite or the improper use of permissible explosives caused 7 of the explosions resulting in the death of 285 men. The poor placement of drill holes, excessively heavy charging of explosive, insufficient or improper tamping of holes, and failure to undercut the coal were some of the reasons for blown-out shots.

The explosives now designated as "permissible" were first used in 1907. Since that time explosives manufacturers have continually improved them, and it is now possible to obtain permissible explosives that, with intelligent use, will give practically as effective blasting results as black blasting powder. Explosives engineers have proved that the production of lump coal does not depend wholly upon the speed of detonation of an explosive and that strength and density, as well as proper placing of holes, are also important factors. The best method of determining the proper permissible explosive to use in any coal mine is to consult the explosives manufacturers, who have engineers well qualified to recommend the proper grade; the tests should be conducted under the constant watchful supervision of some person or persons known to be sympathetic with trying to secure a safe, effective type of explosive.

Under the new New Mexico coal-mining code only permissible explosives fired electrically are permitted, except where shooting is done from the surface and in certain special rockwork.

If permissible explosives are not used according to the methods prescribed by the United States Bureau of Mines and the manufacturers they may ignite an explosive mixture of gas or a coal-dust cloud. Even though the explosive is properly manufactured and approved by the Bureau of Mines it is not permissible used in any of the following ways:

1. If stored under improper conditions until it changes in character.
2. If used frozen or partly frozen.
3. If more than $1\frac{1}{2}$ pounds are used per shot.

Table 1. - Explosions in New Mexico coal mines, 1895 - 1932

Case	Date	County	Type of opening	Tone per year	Number employed	Number in mine	Extent of explosion	Cause of the presence of an explosive atmosphere	Probable source of ignition	Possible means of prevention at time	Gas or dust explosion	Number killed	Number injured	Number escaped
1	2/27/1895	Santa Fe.	Slope.	-	-	-	Entire mine.	-	-	-	-	24	-	-
2	4/21/1899	do.	do.	-	-	-	do.	-	-	-	-	5	-	-
3	3/29/1901	McKinley.	do.	50,000	125	-	do.	Dust raised by shots.	Blown-out shot of black blasting powder.	State mine inspector recommended mine be kept clean and drill holes placed better.	Dust.	3	-	-
4	9/5/1903	Colfax.	Drift.	443,398	307	0	Local.	Gases distilled from burning coal.	Mine fire.	-	Gas.	0	-	-
5	10/5/1906	do.	do.	107,012	78	Unknown.	-	-	Said to be an open light.	-	Gas and dust.	10	-	-
6	12/31/1907	Socorro.	Slope.	63,866	35	do.	Entire mine.	Blown-out shots raised dust cloud.	Blown-out shot of frozen 40 percent dynamite.	-	do.	11	3	2
7	3/17/1910	McKinley.	do.	272,845	210	do.	Local.	Dust raised by shots.	Blown-out shot of black blasting powder.	-	do.	0	2	-
8	10/22/1913	Colfax.	Drift.	-	312	do.	Entire mine.	Dust raised by shots fired in violation of company rules.	Blown-out shot.	State mine inspector recommended mine be kept clean and drill holes placed better.	do.	263	-	24
9	10/23/1918	McKinley.	-	-	-	2	do.	Dust raised by shots.	Blown-out shot and keg of black blasting powder.	Sprinkling, rock-dusting, use of permissible explosive for all blasting.	Gas and dust.	2	0	-
10	4/14/1920	Colfax.	Drift.	-	-	-	do.	do.	Blown-out shot.	Sprinkling, rock-dusting, use of permissible explosive, better tamping, location of holes.	Dust.	5	0	-
11	5/5/1921	Santa Fe.	Slope.	36,327	-	-	Local.	-	-	Improved ventilation.	Gas.	1	-	-
12	11/25/1922	do.	do.	74,352	101	Unknown.	do.	Sudden outflow of gas.	Electric arc from non-permissible electric motor driving a conveyor.	Better ventilation of working places.	do.	12	9	-
13	2/8/1923	Colfax.	Drift.	-	172	122	Entire mine.	Wrecked trip.	Electric arc from trolley.	Sprinkling trips.	Dust.	120	0	2
14	11/15/1930	Santa Fe.	Slope.	50,500	111	111	Local.	Driving drift too far ahead of air.	Electric arc from mining machine.	Use of line brattice to face and permissible mining machine.	Gas.	2	0	-
15	11/17/1930	do.	do.	3,110	15	2	Entire mine.	Dust raised by shots.	Blown-out shot of black blasting powder.	Permissible explosives, rock-dusting and sprinkling dust.	Dust.	1	1	-
16	12/6/1930	do.	do.	46,697	81	81	Local.	Lack of line brattice and shutting off of blower fan.	Arc from dry electric oil switch.	Use of line brattice in place of blower fan.	Gas and dust.	5	7	-
17	12/7/1932	do.	do.	38,683	55	55	do.	Door left open short-circuited air.	Electric arc from 60-volt bell line.	Use of overcast in place of doors.	do.	14	2	39



Table 2. - Number of explosions and men killed and injured
in New Mexico coal mines by years

Year	Number of explosions	Number killed	Number injured	Year	Number of explosions	Number killed	Number injured
1895	1	24	-	1918	1	2	-
1899	1	5	-	1920	1	5	-
1901	1	3	-	1921	1	1	-
1903	1	0	-	1922	1	12	9
1906	1	10	-	1923	1	120	-
1907	1	11	3	1930	3	8	8
1910	1	0	2	1932	1	14	2
1913	1	263	-	Total	17	478	24

Table 3. - Nature of explosions with reference to
gas or coal dust

Nature of explosion	Number of explosions	Number killed	Number injured
Unknown	2	29	0
Gas	4	15	9
Dust	7	400	6
Gas and dust	4	34	9
Total	17	478	24

Table 4. - Sources of ignition

Cause	Number of explosions	Number killed	Number injured
Blown-out shot	7	285	6
Open light	1	10	
Electric	5	153	13
Mine fire	1		
Unknown	3	50	
Total	17	478	24

4. If the diameter of the cartridge is less than that designated in the column "smallest permissible diameter".

5. If fired with an electric detonator of less efficiency than that prescribed or if fired by means of a nonpermissible-type blasting unit.

6. If fired with fuse.

7. If fired without stemming.

8. If fired with combustible stemming.

9. If fired in the presence of a dangerous percentage of fire damp.

10. If the shot is a depending shot, is bored into the solid, or has a burden so heavy that the shot obviously is liable to blow out.

Electricity.- Electricity caused 5 explosions resulting in the death of 153 men. With the increased use of electricity in the coal mines of the United States there has been an alarming increase in the number of gas ignitions and mine explosions from electric arcs.

This condition will continue until the use of nonpermissible electrical equipment is prohibited in any part of the mine not properly rock-dusted and ventilated by a positive current of air which cannot be short-circuited by human error, such as leaving a door or curtain open or improperly installing line brattice.

Where permissible electrical equipment is used and maintained in permissible condition it greatly reduces the hazard of gas ignition in mines that may become gassy.

The Mine Safety Board of the United States Bureau of Mines^{4/} makes the following recommendation regarding electricity in coal mines:

The United States Bureau of Mines recommends that when electricity is used in coal mines rated as gassy, or whenever in any mine the atmosphere may become gassy:

1. Electrical equipment shall be permissible.
2. Nonpermissible electrical equipment shall be used only in pure intake air.
3. Electrical power shall be cut off whenever the air in the workings is in a dangerous condition, due to inflammable gas.

^{4/} Mine Safety Board, Decision 13: Inf. Circ. 6732, Bureau of Mines, 1933, 43 pp.

Operators and mine electricians will find much instructive information regarding electricity in mines in United States Bureau of Mines Technical Paper 402, Safety Rules for Installing and Using Electrical Equipment in Coal Mines.

Mine fires.- The report of the Territorial United States mine inspector gives the cause of the explosion listed as case 4 as fire igniting gases distilled from the heated coal; during the fighting of mine fires in soft-coal mines there is always great hazard of a gas ignition, and men fighting such fires should fully realize this hazard, taking all possible precautions to protect themselves.

Open lights.- Case 5 is the only one in which it is said that an explosion in a New Mexico coal mine was caused by an open light. There have probably been others the records of which are not available to the writer; case 8 is ascribed by some mining men to open-light ignition, although the generally accepted cause is a blown-out shot.

The new (1933) State mining code (section 133) forbids the use of open lights in gassy coal mines or in any mine employing more than 10 men underground. Only permissible safety lamps, either flame or electric, approved by the United States Bureau of Mines shall be used as portable lights in the larger coal mines of New Mexico.

Unknown.- Two explosions are classified as of unknown origin, because the records available are incomplete.

MEANS OF PREVENTING THE EXPLOSION AT THE TIME

Better ventilation.- Better ventilation of working places by use of overcasts in place of doors, line brattice in place of blower fans, or line brattice from the last open crosscut to the face, with a sufficient quantity of air available, would have prevented the accumulation of gas in all the cases listed wherein explosive gas was a factor.

If the recommendations made in the following decision No. 95/ of the Mine Safety Board are strictly adhered to there should be few if any gas explosions:

Bureau of Mines, Department of Commerce, recommends, in coal-mine ventilation practice, the following specifications as to unit quantity and quality of air:

1. The quantity in cubic feet of pure intake air flowing per minute in any ventilating split shall be at least equal to 100 times the number of men in that split.

E/ See footnote 4.

2. The quantity of air entering each unsealed place shall be at least 200 cubic feet per minute and as much more as may be necessary to properly dilute and carry away inflammable or harmful gases which may be present.

3. The air shall be made to circulate continuously to the face in every unsealed place into which an appreciable amount of methane enters.

4. The air in any unsealed place shall be considered unfit for men if it shall be found to contain less than 19 percent oxygen (dry basis), more than 1 percent carbon dioxide, or a harmful amount of poisonous gas.

5. If the air in any unsealed place, when sampled or tested in any part of that place not nearer than 4 feet from the face and 10 inches from the roof, shall be found to contain--

(a) More than $1\frac{1}{2}$ percent of inflammable gas, the place shall be considered to be in hazardous condition and require improved ventilation; and

(b) If more than $2\frac{1}{2}$ percent of inflammable gas is found, the place shall be considered dangerous and only men who have been officially designated to improve the ventilation and are properly protected shall remain in or enter said place.

6. If the air in the split which ventilates any group of workings contains more than $1\frac{1}{2}$ percent of inflammable gas, these workings shall be considered to be in a dangerous condition and only men who have been officially designated to improve the ventilation and are properly protected shall remain in or enter said workings.

Rock-dusting.— Some New Mexico coal mines were among the first in the West to use rock-dust to prevent coal-dust from being a factor in explosions, and several are now among the best rock-dusted mines in the United States.

In at least two explosions in Santa Fe County the records indicate that the explosions were localized by rock-dust. The localization of these explosions is thought to have resulted in the saving of nearly a hundred lives; this fact was strongly stressed by the newspapers and the State coal-mine inspector's annual reports. Even in view of these published statements there were, until the passage of the recent mining code, numerous coal mines in the State that did not use rock-dust in any form.

The State coal mining code of 1933 (article 23) requires thorough rock-dusting of all mines, except those producing pure anthracite, unless all coal dust on floor, ribs, and roof and timber is muddy at all times. This code also specifies the type of dust that may be used, the amount, the

method and time of sampling, and the method of recording the results of analyses of dust samples. Rock-dust barriers are also required, and the amount of dust in each barrier is specified.

Records indicate that as far back as 1894 sprinkling was practiced in Colorado to prevent participation of coal dust in explosions. At present one mine in New Mexico has water piped to every working face; the company operating this mine strictly enforces a rule requiring every miner to keep his place thoroughly wetted down. Several mines in the State sprinkle trips at sectional and main partings. The Bureau of Mines makes the following recommendations^{6/} regarding sprinkling in coal mines:

In the interest of safety in coal mining, the United States Bureau of Mines to lessen the coal-dust hazard recommends that:

1. Machine coal cuttings be wet as the cutting is being done.
2. The coal face, and the working place 40 feet therefrom, shall be kept free of coal-dust by the use of water.
3. The top of loaded cars in the working place shall be wet.

Coal-dust is inevitably produced in the cutting, blasting, and loading of coal. The above recommendations are aimed at wetting, so far as practicable, the dust at the point of its formation in the working places and thus prevent it from rising, as a dry dust will do, into the air and being carried by the air current into adjacent places.

Recommendations 2 and 3, advocating wetting the coal face and tops of loaded cars, can best be effected by having water lines extending to each face and watering by hose, each face region being kept supplied with suitable hose.

These recommendations do not apply to anthracite mines the dust of which will not propagate an explosion.

Table 5. - Causes of gas accumulations or coal-dust clouds

Cause	No. of explosions	No. killed	No. injured
1. Air short-circuited through doors or curtains	1	14	2
2. Use of mechanical blower in place of line brattice	1	5	7
3. Driving drift too far ahead of last open crosscut	1	2	-
4. Sudden outflow of gas	1	12	9
5. Coal-dust cloud raised by shots	7	285	6
6. Coal-dust cloud raised by wrecked trip	1	120	-
7. Gas distilled from coal by mine fire	1	-	-
8. Unknown	4	40	-
Total	17	478	24

6/ See footnote 4.

Air short-circuited through doors.— It is almost impossible to operate a coal mine without some doors, but they should be in pairs or supplemented with curtains; they should be self-closing, and the use of any device to hold them open should be forbidden. In certain New Mexico mines doors can very readily be replaced with overcasts; changes in mining methods and additional air may be required, but the gain in safety will more than justify them.

Use of mechanical blowers in place of line brattice.— Because of the number of explosions that have been caused directly or indirectly by mechanical blowers in coal mines and the other hazards inherent in the use of this equipment the United States Bureau of Mines makes the following recommendation^{7/} regarding blowers in coal mines:

In the interest of safety, the Bureau of Mines, Department of Commerce, recommends that auxiliary fans or blowers should not be used in coal mines as a substitute for methods of regular and continuous coursing of the air to every face of the mine.

Driving entries too far ahead of the last open crosscut.— The present New Mexico coal-mining law (section 140) provides that

In gassy mines, entry and room faces shall be ventilated by a current of moving air directly by a line brattice if necessary so as to keep such faces free at all times from gas as determined by a flame safety lamp. Crosscuts in all coal mines shall be driven not more than one hundred feet apart and such lesser distance apart as is necessary to comply with this provision or as ordered by the state mine inspector. In nongassy mines crosscuts shall not be more than one hundred feet apart unless greater distance is authorized by the state mine inspector. No entry or room shall be abandoned until a crosscut has been made at the face.

If this section is strictly followed the probability of an accumulation of explosive gas at working places will be reduced to a minimum.

Sudden outflow of gas.— This hazard can ordinarily be handled by use of permissible equipment, by adequate sealing of abandoned workings, and by proper ventilation. The alertness and knowledge of the mine officials are determining factors in avoiding hazards of this nature.

Coal-dust cloud raised by shots.— Improper location of drill holes, the use of an insufficient number of holes, shooting off the solid, firing of dependent shots, use of combustible or too little or no stemming, and loading holes too heavily cause blown-out shots. These can be eliminated by proper supervision and by education of miners and shot-firers in safe methods. Also, if the shots are loaded and fired when the working shift is out of the mine the likelihood of wholesale fatalities from explosions caused by blown-out shots or from other causes in connection with the use or abuse of explosives in coal mines will be eliminated.

^{7/} See footnote 4.

One mine in New Mexico fires all shots from the outside with every man out of the mine; the explosives are taken into the mine by the shot firers after the shift is out of the mine, and they load and tamp all holes. The next best method is to have a shot firer load, tamp, and fire all the holes with all others out of the mine. A good shot firer who is kept under adequate supervision will not shoot an improperly drilled hole, a place that is not properly undercut, or a place in which the "bug dust" is not removed from the cut; neither will he overload a hole or tamp it with stemming containing coal dust.

The Bureau of Mines recommends that for blasting in coal mines permissible explosive, fired electrically, be exclusively used; and that, as an aid to blasting, all coal which it is feasible to cut should be cut or sheared. (U. S. Bureau of Mines Safety Board decision 2.)^{8/}

Rock-dusting of all places to within a short distance of the working face and thorough wetting of the places before shooting will lessen the likelihood of occurrence of an explosive coal-dust cloud.

Coal-dust clouds raised by wrecked trip.- Properly maintained and inspected haulage equipment and haulageways will aid in preventing wrecks. Sprinkling of trips, thorough rock-dusting of haulageways, and keeping haulageways clean will prevent dust clouds from being raised by a wreck, or if a dust cloud should be ignited the resulting explosion will be localized.

Gas distilled from heated coal at time of a mine fire.- This phenomenon is illustrated by explosions that often occur in coal stoves. In mines, explosions from mine fires are often very violent due to the large amount of inflammable gas and coal-dust present.

A mining system should be so designed that fires can be readily sealed without danger to the operation of the rest of the mine.

EXPLOSIBILITY OF NEW MEXICO COAL DUST

United States Bureau of Mines tests show that semianthracite, bituminous, and subbituminous coal dust is explosive under certain conditions. Anthracite dust is not thought to be explosive under ordinary conditions.

The presence of even a small quantity of inflammable gas in the mine air increases the explosibility of the dust; an additional quantity of rock-dust must, therefore, be added to neutralize the effect of the gas present, but of course rock-dust will not prevent an explosion when gas is present in explosive proportions (roughly 5 to 15 percent).

^{8/} See footnote 4.

COAL-MINING CODE OF NEW MEXICO

In 1933 through the joint efforts of various mine operators, the Governor, and the State coal-mine inspector, a new code of mining laws was drawn up with specific requirements as to ventilation, rock-dusting, and other explosion-prevention measures. Several operators had these measures in effect long before the passage of the law.

CONCLUSIONS

New Mexico has had a particularly bad mine-explosion record in the past due to the fact that there have been two very destructive explosions with great loss of life and 15 others with considerable loss of life. These disastrous explosions have occurred in a State employing relatively few men in the coal-mining industry compared with other larger coal-producing States. The explosion-prevention measures introduced in the last decade by several operators have virtually eliminated the danger of a general coal-dust explosion in their mines; if other operators carry out similar measures as provided in the new State law, New Mexico should be able not only to eliminate disastrous coal-mine explosions completely but also to reduce the coal-mine accident rate so that it will compare favorably with rates for other States.

INFORMATION CIRCULAR

UNITED STATES BUREAU OF MINES

MINE EXPLOSIONS AND FIRES IN THE UNITED STATES
DURING THE FISCAL YEAR ENDED JUNE 30, 1933^{1/}

By D. Harrington^{2/} and W. J. Fene^{3/}

Explosions are a type of mine accident that can be prevented by known methods; however, they continue to occur because some mining people are still negligent in adopting and using the known and, in general, readily available measures for avoiding these disasters. Although the frequency and severity of explosions have decreased during the past 5 years there is still much room for improvement. Pennsylvania is an excellent example of what may be done to prevent explosions; this State had an explosion record by no means good during the decade before 1930, but by the adoption of readily available and not very expensive safety measures it now has almost entirely eliminated explosions. Pennsylvania bituminous mines now have the enviable record of not having had a major explosion disaster in considerably over 4 years. In fact, many of our mining States are showing definite progress in working mines without explosions; and some of these, like Alabama, West Virginia, Wyoming, and others now operating with few if any explosions, have had anything but a favorable record in the past.

SUMMARY OF MINE EXPLOSIONS BY STATES

Table 1 shows the results of studies of 22 explosions in 10 States, with a total of 122 deaths, during the fiscal year ended June 30, 1933. The number of ignitions (22) is fewer than that (33) for the previous fiscal year; however, the number of deaths (122) is considerably larger than that (87) for the previous year, the increase being due chiefly to one explosion in Illinois which cost 54 lives.

The States with the greatest number of deaths from explosions during the year were Illinois, 54; Kentucky, 26; Pennsylvania, 9 (6 in anthracite mines); West Virginia, 9; Indiana, 4; Oklahoma, 3; Washington, 2; and Colorado, 1. The greatest number of ignitions listed, 6 (5 in anthracite

^{1/} The Bureau of Mines will welcome reprinting of this paper, provided the following footnote acknowledgment is used: "Reprinted from U. S. Bureau of Mines Information Circular 6761."

^{2/} Chief engineer, safety division, and chief, health and safety branch, U. S. Bureau of Mines.

^{3/} Associate mining engineer, U. S. Bureau of Mines Safety Station, Pittsburgh, Pa.

Table 1. - Summary of mine explosions by States, July 1, 1932 to June 30, 1933

	Lighting used			Fatalities by ignition causes									
	Open	Closed	Unknown or no lights	Total mine explosions	Electricity		Open lights		Explosives		Unknown		Total
					Deaths	Igni- tions	Deaths	Igni- tions	Deaths	Igni- tions	Deaths	Igni- tions	
California	1/1	-	-	1/1	-	-	-	1	-	-	-	-	1
Colorado	-	1	-	1	1	1	54	-	-	-	-	1	1
Illinois	1	-	-	1	-	-	-	1	-	-	-	54	1
Indiana	2	-	-	2	-	-	4	2	-	-	-	4	2
Kentucky	2	-	-	2	-	-	23	1	-	-	-	26	2
New Mexico	-	1	-	1	14	1	-	-	-	-	-	14	1
Oklahoma	1	1	-	2	-	-	3	2	-	-	-	3	2
Pennsylvania	-	2/6	-	6	-	1	-	3	-	-	5	9	6
Washington	-	1	-	1	2	1	-	-	-	-	-	2	1
West Virginia	-	4	-	5	7	4	-	-	-	-	-	9	5
Total United States, fiscal year ended June 30, 1933	7	14	1	22	25	8	63	8	4	29	5	122	22
Total United States, fiscal year ended June 30, 1932	1/15	1/15	1/4	1/34	11	10	33	15	5	42	1	87	33
Total United States, fiscal year ended June 30, 1931	12	15	0	1/26	143	9	56	11	6	18	0	217	26
Total United States, fiscal year ended June 30, 1930	8	25	1	34	154	18	21	8	6	24	0	199	34
Total United States, fiscal year ended June 30, 1929	18	19	1	38	93	18	14	11	8	26	6	139	38

1/ Ignition of methane in a metal mine.

2/ Mixed lights used in one mine.

mines), was for Pennsylvania; West Virginia was next with 5 ignitions; Indiana, Kentucky, and Oklahoma had 2 each; and California, Colorado, Illinois, New Mexico, and Washington had 1 each.

Three major explosions (in which 5 or more men were killed) occurred during the fiscal year, all of them in December 1932; 91 persons were killed. The fact that there were only 8 gas or dust ignitions of electrical origin in the mines of the United States during the past fiscal year compared with an average of 14 for the previous 4 years indicates that greater precautionary measures than in the past have been taken with ventilation and in the installation and use of electrical equipment in mines.

Twenty-five persons were killed during the past fiscal year in mine explosions caused by electricity; this is a much poorer showing than that of the preceding year, when only 11 were killed from this cause; however, each of the last 2 fiscal years shows decided improvement in deaths from explosions of electrical origin compared with the 3 previous fiscal years, in which 143, 154, and 93 persons were killed from mine explosions started by electricity. This indicates a 75 percent reduction in deaths from explosions of electrical origin over the average for the 4 fiscal years 1929 to 1932, inclusive, which denotes not only increased precautions in the use of electricity underground but also that rock-dusting is now performing a very useful function in our coal mines in helping to prevent dust ignitions and in stopping gas explosions from picking up inflammable dust to extend or propagate explosions.

West Virginia had the greatest number of electrical ignitions (4), with 7 fatalities; New Mexico had 1 electrical ignition, with 14 fatalities; Washington had 1 electrical ignition, with 2 fatalities; and Colorado and Pennsylvania each had 1 electrical ignition, with 1 fatality.

Eight ignitions were caused by open lights (1 in a metal mine, burning 2 men) compared with 15 in the previous fiscal year and an average of 14 during the previous 4 fiscal years. The deaths last year numbered 64 (54 in one explosion) compared with 33 the previous year and an average of 31 during the previous 4 years. Ignitions from open lights were distributed by States as follows: Illinois, 1, with 54 deaths; Pennsylvania, 3, with 3 deaths; Oklahoma, 2, with 3 deaths; Kentucky, 1, with 3 deaths; and California, 1, with no fatalities. Explosions originated by blasting with explosives numbered 4, with 29 fatalities, compared with 5 ignitions from explosives, with 42 fatalities, for the previous year and an average of 6 ignitions and 30 fatalities for the previous 4 years. One ignition from explosives in Kentucky resulted in the death of 23 persons, while 2 ignitions in Indiana killed 4 and 1 ignition in West Virginia killed 2.

Two explosions in which the cause of ignition could not be determined occurred in Pennsylvania, resulting in the death of 5 persons.

1. The first part of the paper is devoted to a general discussion of the problem of the existence of solutions of the system of equations (1) under the conditions (2). It is shown that the system (1) has a solution if and only if the conditions (2) are satisfied.

2. In the second part of the paper, the problem of the construction of the solution of the system (1) is solved. It is shown that the solution of the system (1) can be constructed by the method of successive approximations.

3. In the third part of the paper, the problem of the stability of the solution of the system (1) is solved. It is shown that the solution of the system (1) is stable if and only if the conditions (2) are satisfied.

RECORD OF EXPLOSIONS BY STATES

Table 2 shows the record of explosions by States during the past 5 fiscal years. During the 5-year period 153 ignitions caused 764 deaths, an average of 30.6 ignitions and 152.8 deaths per year; this tabulation lists only the explosions with which the safety division of the United States Bureau of Mines had contacts of some kind. Both the ignitions and deaths during the past year were fewer than the average, but the number of deaths increased considerably over the previous year. For the 5-year period, July 1, 1928 to June 30, 1933, Pennsylvania heads the list both in total number of ignitions (36) and in deaths (125); Oklahoma ranks second in number of deaths (120), with only 9 ignitions. It is of interest to note that during the 5 years Kansas and Maryland had 1 ignition each, with a total of 3 deaths (2 in Kansas and 1 in Maryland); Tennessee had 3 ignitions, without a fatality, and Oklahoma went 1 year without a single ignition and another year with only 2 ignitions and no deaths. The following coal-mining States and territories are not listed in table 2 as having had an explosion or explosions during the past 5 fiscal years: Alaska, Arkansas, Iowa, Michigan, Missouri, Montana, North Dakota, Texas, and Wyoming. Although this does not mean necessarily that no explosions occurred in any of these States it means that the safety divisions of the United States Bureau of Mines has had no contact with explosions there during the period; on the other hand, the fact that these States do not appear in table 2 does indicate that no explosions with heavy loss either of life or of property occurred in any of them during the past 5 fiscal years.

CAUSES OF EXPLOSIONS

Table 3 gives data as to causes of explosions in the mines of the United States during the past 5 fiscal years, as reported to the safety division of the United States Bureau of Mines. It will be noted that 153 explosions were studied during the 5-year period, an average of about 31 per year. There were 22 explosions during the fiscal year ended June 30, 1933, which is considerably below the average. Sixty-three (41.2 percent) of these explosions of the past 5 years were of electrical origin, 53 (34.6 percent) were caused by explosives, and 8 (5.2 percent) were of unknown origin. Explosions of electrical origin caused 36.4 percent of the explosions during the past year, an increase over the 2 preceding years but a considerable decrease over the first 2 years of the period and also lower than the average of 41.2 percent for the entire 5-year period; hence it is probable that much of the laxity in the use of electrical equipment in coal mines, which was alarmingly evident from 1925 to 1930, has now been corrected.

For the 5-year period, 34.6 percent of the mine explosions in the United States had open lights or smoking as the ignition cause; 36.4 percent were caused by open lights (none by smoking) during the past year, which is slightly above the average for the 5-year period. When the efficiency of the available types of closed lights is compared with that of available open lights, disregarding the numerous important safety features of closed lights, it is difficult to understand why the use of open lights in mines is continued or allowed. Smoking underground should be prohibited in all mines, and this can be done and in many mines is being done by supervision and discipline of the right kind.

Table 2. - Record of explosions by States during last 5 fiscal years

State	1928-29		1929-30		1930-31		1931-32		1932-33		5 year totals		Averages per year	
	Deaths	Ignitions	Deaths	Ignitions	Deaths	Ignitions	Deaths	Ignitions	Deaths	Ignitions	Deaths	Ignitions	Deaths	Ignitions
Alabama	13	7	8	4	0	2	4	5	-	-	25	18	5	3.6
California	0	1	7	2	12	1	0	1	0	1	19	6	3.8	1.2
Colorado	-	-	0	1	-	-	4	3	1	1	5	5	1.0	1.0
Illinois	0	3	10	2	2	1	5	3	4	1	69	10	13.8	2.0
Indiana	-	-	-	-	28	1	4	-	2	2	32	3	6.4	.6
Kansas	2	1	-	-	-	-	-	-	-	-	2	1	.4	.2
Kentucky	11	3	16	1	-	-	3	1	26	2	56	7	11.2	1.4
Maryland	-	-	-	-	-	-	1	1	-	-	1	1	.2	.2
New Mexico	-	-	-	-	-	-	-	-	14	1	22	4	4.4	.8
Ohio	3	3	1	2	88	3	-	-	-	-	92	8	18.4	1.6
Oklahoma	1	1	71	4	45	2	3	-	3	2	120	9	24.0	1.8
Pennsylvania	70	9	16	8	23	7	9	6	9	6	125	36	25.0	7.2
Tennessee	-	-	-	-	-	-	7	3	-	-	0	3	.0	.6
Utah	1	2	30	3	-	-	0	3	-	-	31	5	6.2	1.0
Virginia	2	1	-	-	-	-	5	3	-	-	56	4	11.2	.8
Washington	-	-	17	2	0	2	-	-	2	1	19	5	3.8	1.0
West Virginia	36	7	23	5	11	4	11	7	9	5	90	28	18.0	5.6
Total	139	38	199	34	217	26	87	33	122	22	764	153	152.8	30.6

Table 3. - Comparison of explosion-ignition causes,
fiscal years 1928-29 to 1932-33,
inclusive

Years	Ignition by electricity		Ignition by open lights or smoking		Ignition by explosives		Ignition by unknown causes		Total	
	Number	Percent	Number	Percent	Number	Percent	Number	Percent	Number	Percent
1928-29	18	47.4	11	29.0	8	21.0	1	2.6	38	100
1929-30	18	52.5	8	23.5	6	18.0	2	6.0	34	100
1930-31	9	34.6	11	42.3	6	23.1	0	.0	26	100
1931-32	10	30.1	15	45.5	5	15.2	3	9.2	33	100
1932-33	8	36.4	8	36.4	4	18.1	2	9.1	22	100
Total (5 year)	63	41.2	53	34.6	29	19.0	8	5.2	153	100

During the past 5 years 19 percent of the explosions--as studied by the safety division of the United States Bureau of Mines--were caused by explosives; the number during the past year was 18.1 percent of the total for that year. Unquestionably there have been numerous explosions due to explosives in the coal mines of the United States during the past 5 years in addition to the 29 listed in table 3 that are not recorded in the table, as the representatives of the safety division do not come in contact with many shot-firing explosions, more especially those occurring on the "off shift" in coal mines, in which no life is lost or only 1 or 2 victims are caught. Nevertheless, explosives practice in the coal mines of the United States has been safeguarded much in late years by extending the use of permissible explosives, by substituting electrical methods for those that are less safe, by restricting blasting to the off-shift or to the end of the shift, and by other progressive changes in blasting practice; it will be unfortunate if the excellent results attained by these changes are lost by the present-day tendency toward reintroduction of blasting during the working shift, with the manifest possibilities for multiple deaths of underground workers if an explosion does occur.

EXPLOSION FATALITIES BY CAUSES

Table 4 compares explosion fatalities by ignition causes as studied by the safety division of the United States Bureau of Mines and shows that during the past 5 fiscal years 764 fatalities occurred from explosions in the mines of the United States, an average of 153 per year. The record for the past year, 122 killed, is better than the average for the 5-year period; however, it is worse than the record of the preceding year, which was the best fiscal-year record of occurrence of fatalities from explosions in the mines of the United States in the present century.

Explosions of electrical origin were responsible for 55.7 percent of the deaths in our mines during the past 5 fiscal years; however, during the past year only 20.5 percent of the deaths were due to this cause. The deaths from this cause were considerably reduced during the past 2 years (25 in 1933 and 11 in 1932) compared with those in the first 3 years of this period (143 in 1931, 154 in 1930, and 93 in 1929).

That an average of 85 persons has been killed annually in our coal mines during the past 5 years because of explosions started by electricity is a record of which our mining and machinery industrial leaders should be anything but proud. The decrease in this type of mine accident during the past 2 fiscal years is gratifying, but there is cause for real anxiety as to what may transpire in the fall and winter of 1933-34, when apparently there will be a material increase in coal-mining activity, possibly with considerable carelessness or recklessness in the use of underground electrical equipment and devices. Our coal-mining people are confronted with a real problem in holding explosions to a minimum when or if this increased activity materializes; however, there is no question but that by the use of the right kind and the right amount of continually applied effort, explosions of electrical origin as well as from all other causes can be prevented.

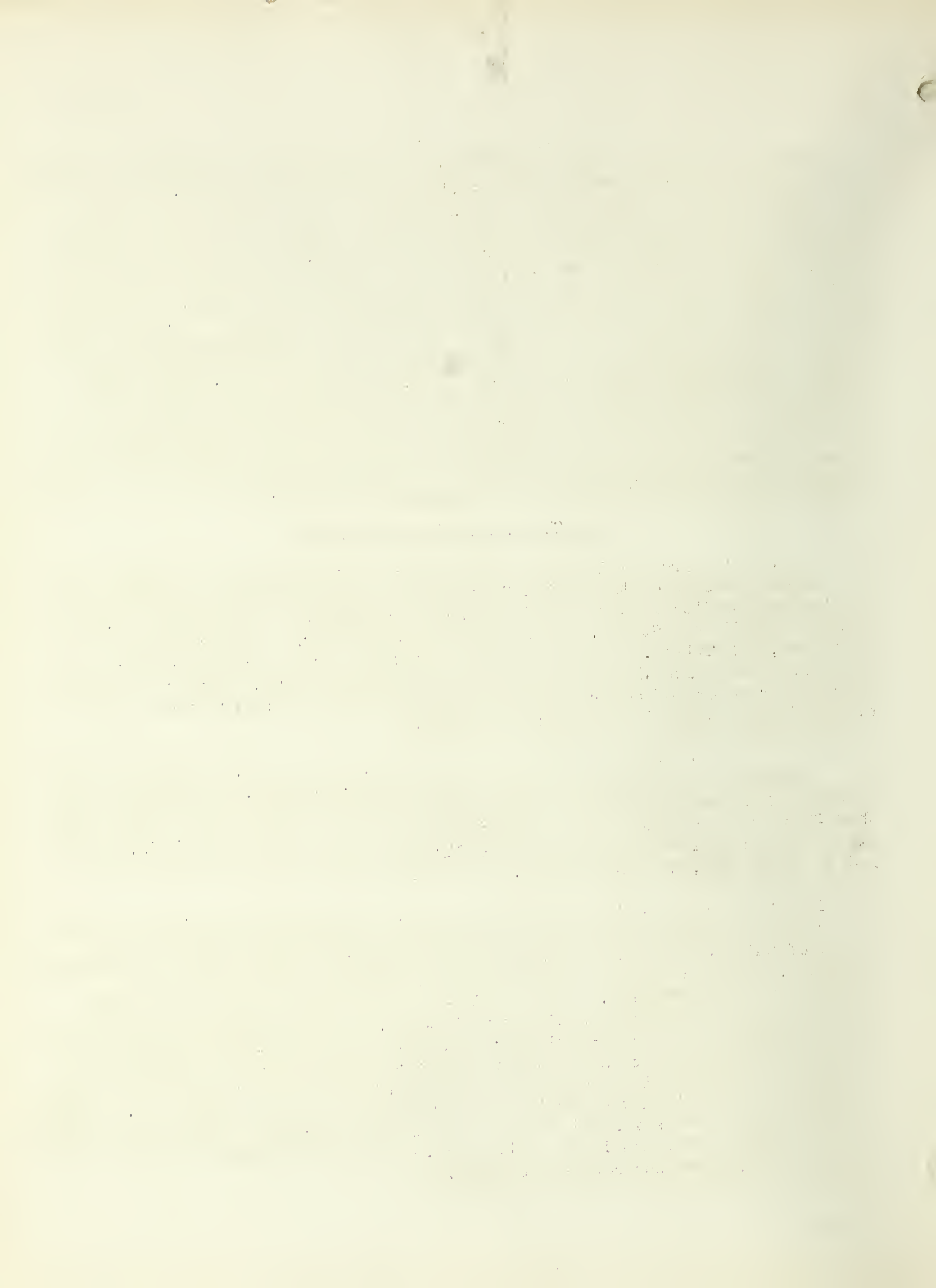
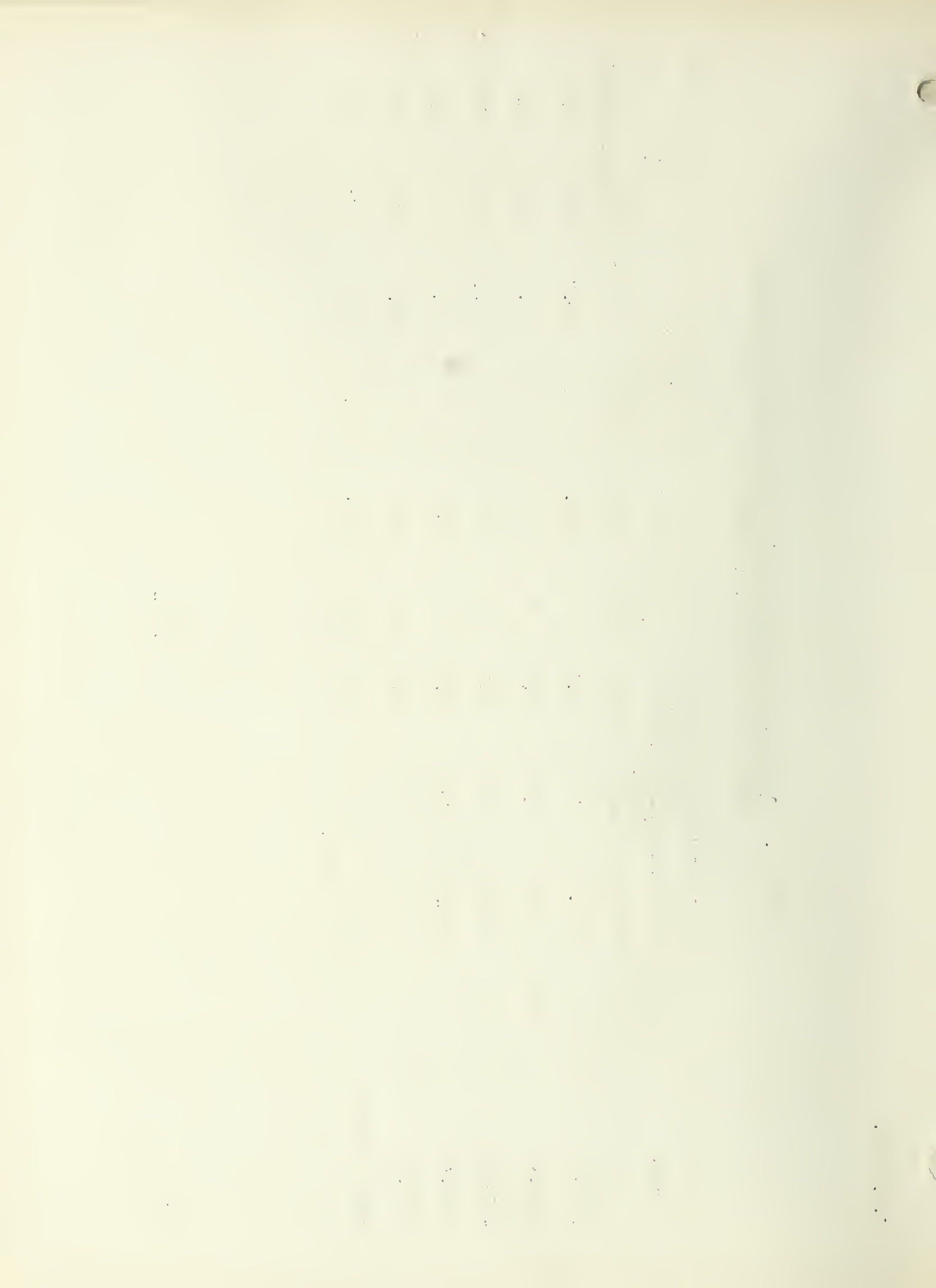


Table 4. - Comparison of explosion fatalities by ignition causes,
fiscal years 1927-28 to 1932-33

Year	Electricity		Open lights		Explosives		Unknown		Total	
	Number killed	Percent	Number killed	Percent	Number killed	Percent	Number killed	Percent	Number killed	Percent
1928-29	93	66.9	14	10.1	26	18.7	6	4.3	139	100
1929-30	154	77.5	21	10.5	24	12.0	0	.0	199	100
1930-31	143	65.9	56	25.8	18	8.3	0	.0	217	100
1931-32	11	12.7	33	38.0	42	48.2	1	1.1	87	100
1932-33	25	20.5	63	51.6	29	23.8	5	4.1	122	100
Total (5 years)	426	55.7	187	24.5	139	18.2	12	1.6	764	100



Open lights (or smoking) caused explosions resulting in 187 fatalities during the past 5 fiscal years or 24.5 percent of the total fatalities from explosions during that period. During the past year there were 63 deaths from this cause or 51.6 percent of the total (54 in one explosion), which is considerably more than in any other year of the past 5. Three of the explosions during the past year were charged to smoking, all in anthracite mines.

It is disquieting to learn that while only 19 persons were killed in explosions started by open lights or smoking in the mines of the United States in the fiscal year ended June 30, 1929 and only 21 in the year ended June 30, 1930 there were 56 in 1931, 33 in 1932, and 63 in 1933, apparently indicating a decidedly upward trend in deaths due to explosions caused by open lights or smoking. The fact that every one of these deaths is utterly unnecessary makes the situation the more deplorable; every underground worker in the United States can and should be supplied with an up-to-date permissible electric cap lamp, which gives better and more dependable illumination than open lights; in addition, there is not an instance available of the loss of even one life from an explosion or a fire started in any mine in the United States by a permissible electric cap lamp, although the records show hundreds killed in one manner or another from the use of open lights.

During the past 5 fiscal years 139 persons have been killed in explosions initiated by explosives, or 18.2 percent of the total killed during this period from all causes. During the past year 29 persons (23.8 percent of the total killed in mine explosions in the United States) were killed in explosions started by blasting or explosives; 23 of these deaths resulted from one explosion caused by dust ignited by a blown-out shot.

The average number killed annually in explosions started by blasting or explosives in our mines during the past 5 years--23--has been exceeded in only 2 years, 1932-33 with 29 and 1931-32 with 42. As previously indicated, the record for the past 5 years (even including the past 2 years) is a vast improvement over what transpired in our coal mines 10, 15, 20, or more years ago. The tendency now is toward extension of the use of explosives and of blasting in mines while the working shift is present; this is unfortunate, and there is reason for the feeling of apprehension now entertained by many, if not most, coal-mine safety men that the increased use of explosives and of blasting during the working shift is virtually certain to increase explosions as well as individual accidents in our coal mines, notwithstanding alleged precautions as to devices, methods, and equipment.

During the past 5 fiscal years 12 persons have been killed in 8 explosions in mines of the United States, and the cause of the explosions has not been ascertained in so far as data could be obtained by the safety division of the United States Bureau of Mines.

ELECTRICAL CAUSES OF EXPLOSIONS

Table 5 gives data relating to causes of explosions of electrical origin in mines of the United States during the past 5 fiscal years in so far as data are available to the safety division of the United States Bureau of Mines. The ignitions by electricity totaled 63 for the 5 years, with 8 causing 25 fatalities during the past fiscal year. The greatest source of electrical ignition in coal mines is from trolley or cable-reel locomotives and from non-permissible mining machines; these two sources, plus trolley wire, were responsible for more than one half of the electrical ignitions during the past 5-year period.

During 1932 trolley or cable-reel locomotives were responsible for 4 ignitions, resulting in 6 fatalities and 3 injuries; a nonpermissible mining machine ignited gas in one mine, killing 2 men and severely burning 4 others; an arc from the controller of a nonpermissible underground-slope hoist ignited gas, killing 1 man and injuring 1 other; an arc from a signal wire on a slope caused an explosion that killed 14 persons; and 2 men were killed by an explosion, the exact source of the electric arc not being determined.

As is true in most electrical ignitions there was evidence of carelessness in maintaining good ventilation in the places where the electrical ignitions occurred during the past 5 years, as well as during 1932-33. In several cases proper inspection of the places had not been made; and in some instances working faces were driven too far ahead of the last open crosscut without proper provision for keeping air circulation at or near the face, and the use of haulageways as return airways, always hazardous, resulted in several explosions.

In one case a mine foreman and a fire boss entered a mine on a trolley locomotive to make an inspection shortly after starting the fan, which had not been in operation for about 100 hours. When they had reached a point about 7,600 feet in by a circuit breaker was thrown into contact, and evidently an arc was formed somewhere along the trolley line in a section of the mine, igniting the gas that had accumulated while the fan was idle. The use of trolley locomotives by fire bosses is dangerous, especially after the fan has been stopped for a considerable period, and this is by no means the only instance in which this practice has resulted disastrously. The use of haulageways on return air, either in gassy or so-called nongassy mines, is another dangerous practice, even though it is in more or less general use, especially in some districts; haulage by electricity, particularly by trolley or cable-reel locomotive, is at all times a potential source of trouble when on return air in mines known to give off explosive gas.

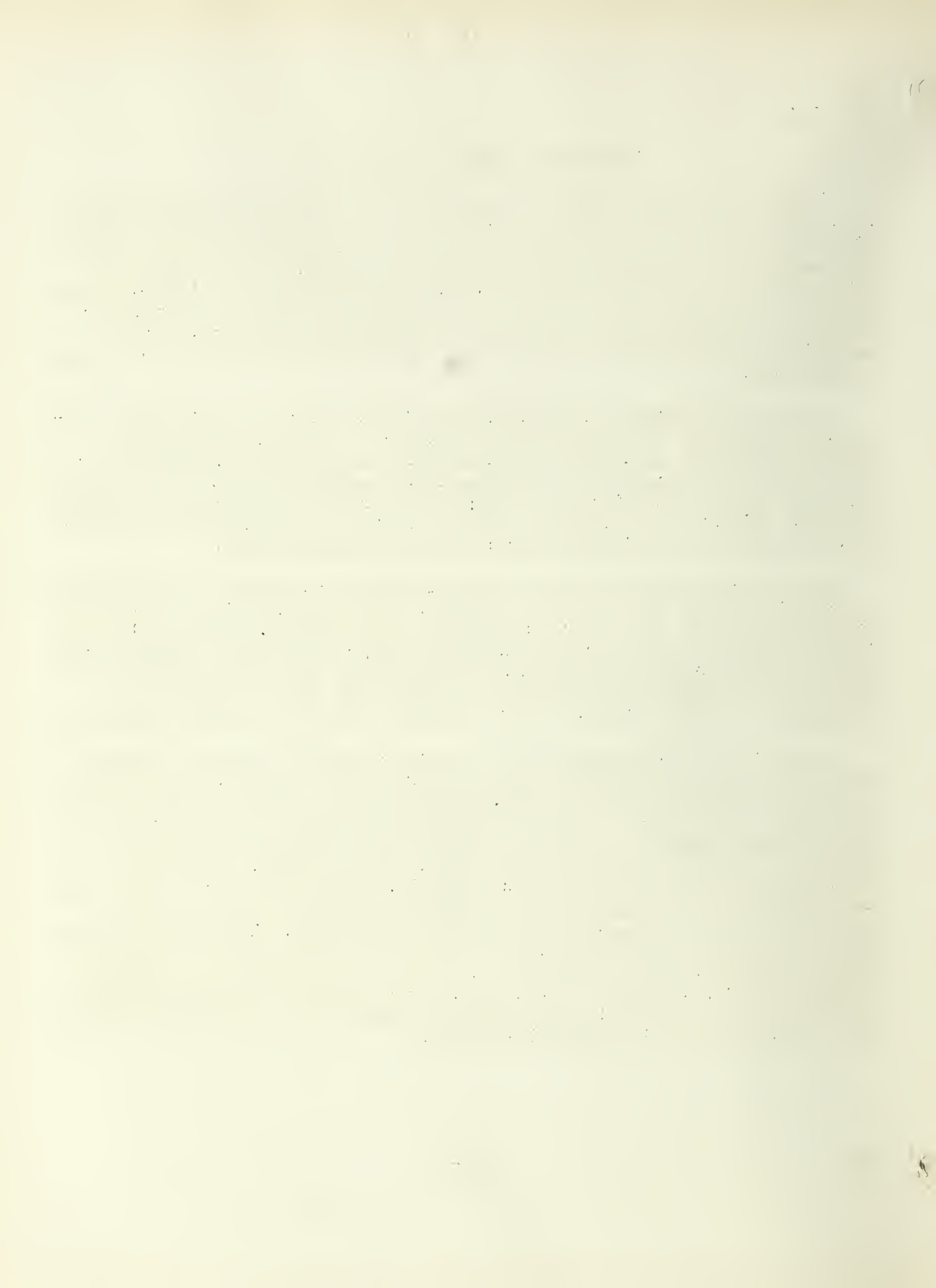


Table 5. - Electrical causes of explosions

Cause	Number of ignitions				Total, 5 years	Number killed		Number injured	
	1928-29 ^{1/}	1929-30 ^{2/}	1930-31 ^{3/}	1931-32 ^{4/}		1931-32	1932-33	1931-32	1932-33
Trolley or cable-reel locomotive	5/5	5	1	1	5/15	0	6	2	3
Nonpermissible mining machine	6/5	5	3	3	6/16	7	2	7	4
Nonpermissible electric motor	4	3	0	0	7	0	0	0	0
Nonpermissible shot-firing battery ...	0	1/2	1/3	1	1	0	0	0	0
Cable nips or cable blowout	0	2	0	0	2	0	0	0	0
Nonpermissible electric drill	0	0	0	2	2	1	0	6	0
Nonpermissible storage-battery locomotive	1	1	2	0	4	0	0	0	0
Nonpermissible electric hoist	0	0	0	0	1	0	1	0	1
Faulty wiring	2	2	0	0	4	0	0	0	0
Trolley wire	1	0	2	1	4	2	0	0	0
Electric arc	2	0	1	2	6	1	2	3	0
Signal wire	0	0	0	0	1	0	14	0	0
Total ignitions	18	18	9	10	63	11	25	18	8

1/ See Inf. Circ. 6178.

2/ See Inf. Circ. 6419.

3/ See Inf. Circ. 6540.

4/ See Inf. Circ. 6680.

5/ 1 not included in totals; may have been from open light, hence is assigned to open lights or smoking.

6/ 1 not included in totals; may have been from smoking, hence is assigned to open lights or smoking.

7/ 1 not included in totals; may have been from overcharged shot, hence is assigned to explosives accidents.

COMPARISON OF LIGHTING IN MINES WHERE EXPLOSIONS
OCCURRED DURING THE PAST 5 YEARS

In a comparison of lighting in mines where explosions occurred during the past 5 fiscal years, as shown in table 6, it is observed that of 155 explosions listed 60 (38.7 percent of the total) occurred in open-light mines, that 88 (56.8 percent) were in closed-light mines, and that in 7 (4.5 percent) the lighting practice was unknown. The fact that the greater number of these explosions occurred in closed-light mines indicates failure to correct other unsafe features, such as inadequate ventilation, use of nonpermissible electrical equipment, use of unsafe explosives, and dangerous methods of employing explosives and electrical equipment.

In most instances closed lights (which as far as general working purposes are concerned in the coal mines of the United States at this time means use of permissible electric cap lamps) have been employed because the management feared the mine had gas or dust conditions such that the use of open lights presented a possible ignition agency; this has resulted in the use of closed lights in most of the highly hazardous mines, where the danger from explosion is greatest. Therefore it might be natural to expect these mines to have the greatest number and percentage of explosions if it were not for the fact that these so-called more dangerous mines can, if they will, protect themselves and their employees from explosions by adopting and keeping in effect readily available measures that will absolutely prevent occurrence of explosions in these mines. Hence the occurrence of explosions in closed-light mines cannot be correctly interpreted as due to the closed lights but as due entirely to neglect or carelessness of some kind in the closed-light mine which has the explosion. There has never been an explosion caused by an approved electric cap lamp, but the introduction of these lamps into some mines has in numerous instances tended to cause relaxation of other common-sense safety precautionary measures.

In one explosion during the past year in a so-called nongassy mine open lights caused a gas ignition, with subsequent explosion, killing 54 men; in another mine known to be gassy open lights caused an explosion in which a surveying party of three was killed. In both explosions the ventilation system was inadequate due to an attempt to operate with a continuous air-current circulation and the supplementary use of booster fans. These practices indicate either lack of knowledge of efficient ventilation methods or wanton disregard of sane and safe practices.

Every year has its quota of explosions started by open lights in so-called nongassy mines, and generally (though not invariably) the occurrence of the explosions leads to the substitution of closed lights for open. Open lights should not be allowed in any mine, coal or non-coal, as they add unnecessary hazards to the already far too numerous dangers that confront the underground worker; and to allow the use of open lights in mines known to give off explosive gas, whether in large or small amount, is little short of criminal negligence when it is realized that ignition of as small a quantity as 150 to 200 cubic feet of an explosive mixture of methane and air under certain conditions can precipitate a widespread explosion which might, probably would, kill every person in the mine.

Table 6. - Comparison of lighting in mines where explosions occurred,
fiscal years 1927-28 to 1932-33, inclusive

Years	Open lights		Closed lights		Lighting unknown		Total	
	Number of explosions	Percent	Number of explosions	Percent	Number of explosions	Percent	Number of explosions	Percent
1928-29	18	47.4	19	50.0	1	2.6	38	100
1929-30	8	23.5	25	73.5	1	3.0	34	100
1930-31	12	44.4	15	55.6	0	.0	1/ 27	100
1931-32	15	44.1	15	44.1	4	11.8	1/ 34	100
1932-33	7	31.8	14	63.6	1	4.6	22	100
Total (5 years)	60	38.7	88	56.8	7	4.5	2/155	100

1/ One mine had mixed (open as well as closed) lighting.

2/ Mines had mixed (open as well as closed) lighting.

Table 7. - Summary of mine fires by States - July 1, 1930 to June 30, 1933

	Fatalities by ignition causes												Total				
	Electricity				Open lights				Explosives						Miscellaneous		
	Open	Closed	Lights	Unknown	Total	Deaths	Igni- tions	Deaths	Igni- tions	Deaths	Igni- tions	Deaths	Igni- tions	Deaths	Igni- tions	Deaths	Igni- tions
Alabama	-	-		1	1	-	1	-	-	-	-	-	-	-	-	-	1
Colorado	1/2	1		-	3	-	-	-	-	-	-	-	-	-	3	-	3
Illinois	1	1		-	2	-	1	-	-	-	-	-	-	-	1	-	2
Indiana	2	-		-	2	-	-	-	-	-	-	-	-	-	-	-	2
Kentucky	3/2	-		2/5	7	-	2	-	-	-	-	2/1	-	-	5	1	7
Missouri	3/1	-		-	1	-	-	-	-	-	-	-	-	-	1	-	1
Montana	3/1	-		-	1	-	-	-	-	-	-	-	-	-	1	-	1
New Mexico	-	1		-	1	-	-	-	-	-	-	-	-	-	1	-	1
Pennsylvania	-	2		3/2	4	-	1	-	-	-	-	-	-	-	3	-	4
South Dakota	1	-		-	1	-	-	-	-	-	-	-	-	-	1	-	1
Tennessee	1	-		-	1	-	-	-	-	-	-	-	-	-	1	1	1
Utah	1/1	-		-	4/1	-	-	-	-	-	-	-	-	-	-	-	1
Virginia	-	-		1	1	-	1	-	-	-	-	-	-	-	-	-	1
West Virginia	-	1		-	1	-	-	-	-	-	-	-	-	-	-	-	1
Total fiscal year, ended																	
June 30, 1933	12	6	9		27	-	7	-	-	-	-	2	5/18	2		2	27
Total fiscal year 5/, ended																	
June 30, 1932	8	5	8		21	1	8	0	0	0	0	0	10	1		1	21
Total fiscal year 1/, ended																	
June 30, 1931	12	7	5		24	0	7	5	3	0		18	9	23		23	24
Total fiscal year 8/, ended																	
June 30, 1930	7	8	9		24	2	9	3	6	0		0	6	5		5	24

1/ Metal mine.

2/ Tiptple fire burned rope, cage dropped and caught 5

men placing rail across top of shaft - 1 killed, 4 injured.

3/ Abandoned mine.

4/ Fire in coal-storage bin near tipple.

5/ Eight of these fires were from spontaneous combustion

6/ See Inf. Circ. 5680, issued January 1933.

7/ See Inf. Circ. 6540, issued December 1931.

8/ See Inf. Circ. 6419, issued January 1931.

Note. - This tabulation includes data only on mine fires to which the attention of the safety division of the Bureau of Mines was called. (1530)

MINE FIRES

Although mine fires have potentialities of causing great loss of life and tremendous property damage the mines of the United States have been fortunate during the past decade in not experiencing a fire causing large loss of life; however, there have been a number of fires that have caused considerable property damage.

During the past fiscal year 27 mine fires--5 in metal mines--were called to the attention of the safety division of the United States Bureau of Mines. Unquestionably, scores of other fires occurred which were detected and extinguished promptly without much damage, hence their occurrence was not made public. Table 7 shows that most of the fires in 1932-33 which were contacted by the safety division forces were in Kentucky, the coal mines of this State being listed as having 7 fires, with 1 fatality; this fatality can be charged only indirectly to a mine fire, as the man was killed by a falling cage when a fire in the tippie burned the rope. Four fires occurred in Pennsylvania, 2 in abandoned mines, which threatened the destruction of much valuable city property. Three fires occurred in Colorado, 1 in a metal mine, 2 each in Illinois and Indiana, and 1 each in Alabama, Missouri (metal mine), Montana (metal mine), New Mexico, South Dakota, Tennessee, Utah (metal mine), Virginia, and West Virginia. One death resulted in a State convict mine when a prisoner set fire to some timbers. It is noted that nearly one third of the fires the causes of which are known were due to electricity, 2 were caused by explosives, and 8 were of spontaneous origin.

SUMMARY

The encouraging factor in connection with explosions is that during the past 5 years the trends in the number of explosions and deaths from explosions have been definitely downward. Although the depressed condition of the coal industry during the past few years may have influenced the downward trends, it is believed that the greatest factors in the reduced frequency and severity of explosions are (1) increased activities of various kinds by State mine-inspection departments, especially along the line of educational work; (2) the rapidly growing realization of the economic waste of all accidents, including explosions, and the more or less general acceptance of responsibility for accident occurrence by the mine operators; and (3) the more widespread dissemination of safety data by the United States Bureau of Mines and other safety organizations.

It is definitely known that mine explosions and fires can be prevented, the problem being chiefly a matter of education and adoption of known preventive measures, the expense of which is negligible when compared with probable (in many instances) actual saving.

Salient points brought out in the study of mine explosions and fires in mines of the United States coming to the attention of the safety division of the United States Bureau of Mines for the year ended June 30, 1933, are as follows:

1. Twenty-two explosions were studied, 16 in bituminous mines, 5 in anthracite mines, and 1 in a metal mine, compared with 33 for the previous year and an average of 30 per year for the past 5 fiscal years.

2. Explosions took 122 lives last year compared with 87 the previous year and an average of 153 annually for the past 5 fiscal years.

3. The 3 major explosions accounted for 91 deaths, while the 19 lesser explosions accounted for 31 deaths. All major explosions (those in which 5 or more persons were killed) occurred in one month (December 1933), hence the remaining 11 months of the fiscal year ended June 30, 1933 were free of major explosion disasters in the mines of the United States, an all-time record of avoidance of major explosion disasters in American mines in so far as records are available.

4. Seven explosions were in open-light mines compared with 15 the year before, an average of 12 annually for the past 5 fiscal years; 14 explosions were in closed-light mines compared with 15 the previous year, an average of 17.6 annually for the past 5 fiscal years. Some of the open-light mines in which explosions occurred were known to give off explosive gas, in some instances in considerable quantities.

5. Eight explosions, causing 25 deaths, were of electrical origin compared with 10 explosions and 11 deaths for the previous year. While the number of explosions caused by electricity decreased the deaths from this cause increased more than 50 percent. Explosions of electrical origin averaged 12.6 in number annually for the past 5 fiscal years, and deaths averaged 85.2 annually for the same period; hence, both of the past 2 fiscal years show a decided improvement in this phase of explosion occurrence in United States mines.

6. West Virginia, with 4 explosions, ranks first in the number of explosions caused by electricity during the year; and New Mexico, with 14 fatalities, ranks first in the number of deaths from this cause.

7. Trolley or cable-reel locomotives were responsible for 4 of the 8 explosions of electrical origin, while a nonpermissible mining machine, a nonpermissible electric hoist, a signal wire, and an electric arc were responsible for 1 each; the explosion caused by an arc from a signal wire caused 14 fatalities.

8. Eight explosions, with 63 deaths, were caused by open lights or smoking, compared with 15 explosions and 33 deaths in the previous year. One of these open-light explosions, which occurred in December 1932, killed 54 persons.

9. Four explosions and 29 deaths were caused by the use (or probably more nearly correctly the misuse) of explosives, compared with 5 explosions and 42 deaths in the previous year.

10. Rock-dust was used to some extent in at least 7 of the mines in which explosions occurred during the last fiscal year. It was effectively applied in only 3 mines; in 2 of the mines in which explosions occurred rock-dust is credited with stopping the spread of the explosions, and it is believed that this saved the lives of many if not all of the approximately 300 persons who were in these mines at the time of the explosions and who escaped essentially uninjured.

11. Eighteen of the explosions during the past year were initiated by gas and 4 by coal dust.

12. The underlying cause of at least 81 of the 122 deaths resulting from explosions during the past year was defective ventilation, and this was manifest in both open and closed-light mines.

13. During 1932-33 the personnel of the safety division of the United States Bureau of Mines reported on 27 mine fires, 22 in coal mines and 5 in metal mines, compared with 21 fires in 1931-32 and 24 in each of the 2 years before that.

14. Nearly two thirds of the fires for the past year, of which the causes are definitely known, were initiated by electricity. Two mine fires were caused by explosions, and at least 8 were of spontaneous origin.

15. The activities of United States Bureau of Mines personnel during the past fiscal year included contacts with 7 mine fires in Kentucky, 4 in Pennsylvania, 3 in Colorado, 2 each in Illinois and Indiana, and 1 in each of the following States: Alabama, Missouri, Montana, New Mexico, South Dakota, Tennessee, Utah, Virginia, and West Virginia.

CONCLUSION

The data in this publication were compiled in the same manner as in previous years (see Inf. Circs. 6173, 6419, 6540, and 6680) and embody only facts brought out during fairly close study by field employees of the safety division of the United States Bureau of Mines. Although an attempt is made to obtain data on every mine explosion that occurs in the United States some of the less destructive explosions are not brought to the attention of the Bureau early enough to allow an investigation to be made; rarely, if ever, however, does the Bureau fail to get fairly complete data on mine explosions in which more than one life is lost or in which any considerable amount of property is damaged. Hence the data in the publications are relatively complete as to mine explosions, especially those causing loss of more than one life. The data on fires are by no means as nearly complete and inclusive as those for explosions, and there is good reason to believe that scores of mine fires (many of them in metal mines) occur annually without having been made known to the United States Bureau of Mines or even the State inspection forces. Mine fires are of almost daily occurrence with some coal-mining companies, particularly at shot-firing time or where the coal fires spontaneously, and fires of electrical origin or from open lights are by no means unusual in metal mines when they are working fairly close to capacity.

The facts obtained from a detailed study of available data on explosions and fires in the mines of the United States during the past 5 or 6 fiscal years are decidedly encouraging to those who year after year despaired of ever having the mines of the United States operated with reasonable freedom from explosion disasters. For the 15-year period prior to 1929 explosions took an average of about 265 lives annually in the coal mines of the United States; and since that date, according to data in tables 1 and 4, the number of fatalities from explosions in the mines of the United States has never been as high as this average figure of 265 per year, being 139 in the fiscal year ended June 30, 1929, 199 in 1930, 217 in 1931, 87 in 1932, and 122 in 1933, the average for the last 5 fiscal years being 153, or very much lower than the 265 for the previous 15 years.

The decreasing number of explosions of electrical origin and of fatalities from them during the past 2 years is especially encouraging, as there had been an alarming increase in explosions of electrical origin and of their very heavy fatality results for several years before 1930 or 1931. It now appears that although mechanization of mines is proceeding (even though at a somewhat reduced rate recently) it is being done with reasonable regard to the hazards involved and apparently with much more attention to overcoming or eliminating those hazards than in 1924 to 1928, inclusive.

The fact that no explosions of gas or dust are listed as having occurred during the past 5 fiscal years in coal mines in Alaska, Arkansas, Iowa, Michigan, Missouri, Montana, North Dakota, Texas, and Wyoming is significant, especially as to Arkansas and Wyoming, both of which have in the past been rather prone to have explosions. Pennsylvania did not have a major explosion disaster in either bituminous or anthracite mines during the past fiscal year, an exceptionally fine performance; in fact, Pennsylvania has not had a major mine explosion since May 29, 1931, or including 2 fiscal years. Pennsylvania, with bituminous mines employing approximately one fourth of the coal miners of the United States and producing about one fourth of the country's coal, much of it from mines with inherently dangerous conditions as to gas and dust, has not had a major explosion disaster (one with 5 or more lives lost) since March 21, 1929, or more than $4\frac{1}{2}$ years at this writing (September 1, 1933); the anthracite mines of Pennsylvania, employing about one fifth of the miners of the United States, many of them in gassy, steeply pitching coal beds, have not had a major explosion disaster since May 29, 1931, or about $2\frac{1}{3}$ years as this is being written. West Virginia, with the number employed and tonnage produced but slightly less than in Pennsylvania bituminous operations and with its numerous gassy and dusty mines, has not had a major explosion since November 3, 1931, or nearly 2 years. Alabama, with numerous hazards in its coal mines, has not had a major explosion since December 28, 1931, or nearly 2 years; during that period only 1 person was killed in Alabama coal mines from ignitions of gas or dust. Colorado, with its dangerous coal-mining conditions and very bad explosion record prior to 1920, has not had a major coal-mine explosion since May 27, 1927, or nearly $6\frac{1}{2}$ years, and has had only two major coal-mine disasters (one with 7 fatalities and one with 6) in the past 10 years. Utah, with the worst fatality rate from coal-mine explosions of any

State in the Union, has not had a major coal-mine explosion since March 8, 1930, and Wyoming, with a bad past as to occurrence of coal-mine explosion disasters, has avoided them since September 16, 1924, or 9 years. Oklahoma has not had a major explosion since October 27, 1930, or nearly 3 years; and Washington has not had one since April 12, 1930, or $3\frac{1}{2}$ years. Thus it appears that at least eight important coal-mining States with bad records as to occurrence of coal-mine disasters have in recent years been relatively free of them, indicating that our coal-mining people at last realize that coal-mine explosions are preventable if readily available methods, devices, and equipment are put into effect and kept in effect.

The three major explosions of the past fiscal year all occurred in December 1932, hence for 11 months of the fiscal year the mines of the United States were free of major disasters, a truly fine record. The calendar year 1933 to date of writing this paper (September 1, 1933) has been free of major explosion disasters in the mines of the United States, and there has not been a major explosion disaster since December 23, 1932, unquestionably the longest continuous period of immunity from major explosion disasters in the available records of mining in the United States.

With the new industrial conditions brought about by the codes of the National Recovery Administration about to go into effect as this is written, there will undoubtedly be numerous changes in mining practice in many States; inasmuch as the codes applied to mining contain essentially nothing with reference to safe operation, unless much extra effort is taken by our mining people toward the safety phases of working our mines there are likely to be numerous relaxations and inefficiencies in our mines, with attendant increased accident rates and probably with increased occurrence of mine disasters. To avoid any such unfortunate condition it is hoped that increased rather than diminished attention will be paid to prevention of accidents, including explosion disasters, in our mines and that at least some attention will be devoted to this important phase of mining when the codes are placed in actual working practice.

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FEBRUARY, 1934

UNITED STATES BUREAU OF MINES
SCOTT TURNER, DIRECTOR

INFORMATION CIRCULAR

JUL 11 1934

OPERATING COAL MINES WITHOUT ACCIDENTS



BY

D. HARRINGTON AND W. J. FENE

INFORMATION CIRCULAR
UNITED STATES BUREAU OF MINES

OPERATING COAL MINES WITHOUT ACCIDENTS¹

By D. Harrington² and W. J. Fene³

Coal mines and mining, as well as metal and nonmetallic mines and mining, are not inherently unsafe, although mining people think they are and would like to have everyone else think so. It is now being demonstrated that mines can be operated with essentially as much safety as any other major industrial enterprise; but most mining people, including workers as well as officials, have been slow to realize or admit this.

In the past, and with most people even at present, the idea has prevailed that it is utterly impossible to operate a mine without accidents. Although, as in all other kinds of industrial activity, it is true that some injury is likely on occasion to occur to some workers and an extraordinary accident may occur to one or more persons under exceptional conditions, mines can be operated with few if any accidents if a reasonable amount of the right kind of precautionary effort is exerted and kept in effect.

After publication of a considerable amount of well-authenticated data indicating that cement plants, some with large numbers of workers, operated nearly full time for a year or more without a lost-time accident and that underground iron-ore mines were also operating for months, many of them for a year or more, without a lost-time accident, a few coal mines began to discard the tradition that accidents absolutely must occur in the mining of coal and began to consider accident occurrence in terms of no lost-time accidents for a month or thereabouts rather than to confine activities entirely to prevention of fatalities. It was not, however, until within the past few years that coal mines actually attempted to operate as long as a year without a lost-time accident or came forward with records well-supported by evidence that coal mines with a fair complement of workers and producing reasonably heavy tonnage could be worked for weeks or months or even for a year or more without a lost-time accident.

1 The Bureau of Mines will welcome reprinting of this article, provided the following footnote acknowledgment is used: "Reprinted from U. S. Bureau of Mines Information Circular 6762".

2 Chief, health and safety branch, U. S. Bureau of Mines, Washington, D. C.

3 Associate mining engineer, safety division, U. S. Bureau of Mines, Pittsburgh Experiment Station, Pittsburgh, Pa.

In 1923 the Portland Cement Association, trying to decrease accident occurrence in its 100 or more plants, including quarries and in some instances underground mines, offered a trophy to any entire plant in the association that would burn clinkers more than 6 months and operate through an entire year--January 1 to December 31, inclusive--without a lost-time accident. Those who formulated the offer were of the opinion that no plant could possibly operate through an entire year without a lost-time accident, but it was expected that in trying to achieve this very difficult goal there would be at least an approach to it and in the effort accidents with their manifold miseries and heavy costs would at least be reduced.

During the first year, 1923, no plant was able to operate without a lost-time accident, although a trophy was given to the plant with the best accident record; but the next year one plant won the trophy, the third year 2 plants won, and each succeeding year greater numbers won until 1931 when 42 plants of 122 in the contest worked through the year without a lost-time accident and 33 won trophies, the other 6 not having operated sufficiently to qualify for receiving a trophy. Hence what was considered an utter impossibility for any cement plant when the safety campaign was started in 1923 was actually accomplished by more than one third of all the plants in the contest in 1931.

By the end of 1932, 12 plants had operated more than 1,000 days each without a lost-time accident and 2 of them--The Iola Plant, Lehigh Portland Cement Co., Iola, Kansas, and the Iron-ton, Ohio, plant of the Alpha Portland Cement Co.--had each worked 6 consecutive years without a lost-time accident. Moreover, 1 cement plant by the end of 1932 had worked an underground mine for more than 6 years without a lost-time accident, and in that period the mine had been in actual operation for more than 1,000 days and had averaged over 1,000 tons of rock produced per day of operation.

Iron-ore mines in the Lake Superior region, as well as some of the lead-zinc properties of the Tri-State district near Picher, Okla., started several years ago to show that mines--both open-pit and underground operations--could be worked long periods, not only without fatal or serious accidents but even without occurrence of lost-time accidents or any accident that would not allow the victim to return to his regular work on the following day.

Although unquestionably the majority of those engaged in both coal mining and metal mining still cling to the idea that accidents of nearly all kinds, including fatalities and occasionally a multiple accident or disaster, are sure to happen as long as mining is done; on the other hand, it is decidedly encouraging to note the number of progressive coal, metal, and nonmetal companies now handling accident-prevention activities along the line of operating long periods without lost-time accidents and feeling--virtually knowing--that with the proper procedure mines can be operated without accidents.

Most coal mines, however, have continued to be operated in the belief (in which practically every one in the industry participated or participates) that coal mining is so inherently hazardous that not only serious accidents

(including permanent partial or permanent total disabilities) but fatalities as well must occur as long as mining is done; that every coal mine should expect one or more fatalities per year; and that the coal mine which escaped disasters, such as fires or explosions, with multiple loss of life, could count itself exceptionally fortunate.

The success or failure of the safety movement depends to a great extent upon the attitude taken by the individual company through its operating executives. Safety in coal mining can only be brought about by unity of mind and purpose of the managing officials and the employees. Proof that coal mining can be conducted without accidents is shown by the experience of 12 individual companies who have operated their mines 1 to 2 years without lost-time accidents. Table 1 shows a record of mines that have operated 1 year or more without a lost-time accident; there may be other mines that have equally as good records as those listed, but their records are not available. It is shown that three mines operated well over one half million man-hours without a lost-time accident and that four mines operated over one quarter million hours. Certain mines were continuing to operate without lost-time accidents after the end of the period shown.

OUTSTANDING SAFETY RECORDS

1. Alloy Mine

The Alloy mine of the Electro-Metallurgical Co. at Alloy, W. Va., operated without a lost-time accident for nearly 2 full years--March 14, 1931 to March 1, 1933--with a production of 234,977 tons of coal, a total of 541,185 man-hours worked by an average of 130 employees.

This is one of the best coal-mining safety records available from the viewpoint of long-time operation and man-hours of exposure without a lost-time accident.

The Electro-Metallurgical Co. is a subsidiary of the Union Carbide Co., New York. The Alloy mine is operating in the No. 2 Gas coal bed, which averages about 4 feet in thickness. The mine is developed by drifts, with the room-and-pillar system of mining, and is more or less in its development stage; it is rated as nongassy. The roof is exceptionally weak over the greater part of the mine and requires heavy and careful timbering. Haulage is mechanical; all coal is undercut and blasted with permissible explosives, and loading is by hand.

Table 1. - Record of mines that have operated 1 year or more without a lost-time accident

Mine	Operator	Place	Period, months	Men em- ployed	Coal mined, tons	Exposure, man- hours
Alloy	Electro-Metallurgical Co.	Alloy, W. Va.	23-1/2	130	234,977	541,185
Dehue	Youngstown Mines Corporation	Dehue, W. Va.	20	214	356,805	531,382
Imboden	Stonega Coke & Coal Co.	Imboden, Va.	15-1/2	198	231,469	509,232
Brier Hill	Buckeye Coal Co.	Brier Hill, Pa.	12	257	184,306	343,363
No. 1	Phelps-Dodge Corporation	Dawson, N. Mex.	12	70	68,237	1/ 12,905
Orenda	Davis Coal & Coke Co.	Boswell, Pa.	13-1/2	180	129,841	279,393
Steubenville	Consumers Mining Co.	Steubenville, Ohio	24	84	128,630	230,732
Hull	DeBardeleben Coal Corporation	Dora, Ala.	2/ 12	160	100,699	301,763
Weyanoke	Weyanoke Coal & Coke Co.	Lowe, W. Va.	12	100	125,000	146,589
Mossboro	Black Diamond Coal Mining Co.	Mossboro, Ala.	15	50	44,436	144,925
No. 1	Block Coal & Coke Co.	Block, Tenn.	13	115	67,832	-
Rockvale No. 3	Colorado Fuel & Iron Co.	Canon City, Colo.	12	60	41,154	-

1/ Man-shifts.

2/ This record actually covered 11 months and 20 days in the calendar year 1930.

2. Dehue Mine

The Dehue mine of the Youngstown Mines Corporation at Dehue, W. Va., operated 602 days without a lost-time accident from January 7, 1931 to August 31, 1932, with an average of 214 men working 531,382 man-hours and producing 356,805 tons of coal.

The mine has consistently maintained a good safety record for a number of years, and the record above is one of the best known from the standpoint of the number of man-hours of exposure and tons of coal produced without a lost-time accident. This mine was awarded certificates of honor by the Joseph A. Holmes Safety Association for meritorious safety records during 1932 and 1933.

The Youngstown Mines Corporation is a subsidiary of the Youngstown Sheet & Tube Co. The Dehue mine is operated through two shafts and two slopes in the Eagle coal bed, which averages about 5 feet in thickness. Development consists of two multiple systems of headings--one of 8 parallel entries and the other of 12. The room-and-pillar method of mining is employed, with pillars recovered on the retreat by machine methods. The roof is fairly good. The mine is classed as gassy; closed lights and permissible explosives are used, and the mine is thoroughly rock-dusted. Haulage is mechanical and all coal is undercut and loaded by hand.

3. Imboden Mine

The Imboden mine of the Stonega Coke & Coal Co., Imboden, W. Va., operated without a lost-time accident from March 20, 1931 to July 8, 1932, or 475 calendar days, with a production of 231,469 tons of coal and a total of 509,232 man-hours of exposure by an average of 198 employees.

The Imboden mine has made remarkable progress in safety by adopting modern safety measures. Two fatalities have occurred since July 18, 1929, with well over 2 million man-hours exposure.

This is a drift mine operating in the Imboden coal bed, which averages 5 feet in thickness and is worked by the room-and-pillar method of mining. Pillars are recovered on the retreat by the open-end method. The immediate roof is a weak slate and requires much timbering. Haulage is mechanical. The mine is rated as nongassy. The coal is centercut and loaded by hand.

4. Brier Hill Mine

The Brier Hill mine of the Buckeye Coal Co., Pine Hill, Pa., operated from December 10, 1930 to December 11, 1931 without a fatality or a disability exceeding the day of the injury, with a production of 184,306 tons of coal by an average of 257 men for a total of 343,363 man-hours.

The mine, operated through a shaft 654 feet deep, is working the Pittsburgh coal bed, which averages 84 inches in thickness. The advance work was by the room-and-pillar method, but all work at the present is retreating,

mining pillars, and stumps. The mine is rated as gassy, and fire bosses are employed. The mine is rock-dusted. Haulage is by trolley locomotives and animals. The roof is bad, and considerable timbering, including forepoling in some places, is necessary. All coal is undercut and loaded by hand. This property is a subsidiary of the Youngstown Sheet & Tube Co., which also is understood to control the Dehue mine.

5. Dawson No. 1

The Dawson No. 1 mine of the Phelps-Dodge Corporation, Dawson, N. Mex., worked the entire year 1931 without a lost-time accident. During this period 70 underground employees worked 12,905 man-hour shifts and produced 68,237 tons of coal. During 1930 this mine had only 4 lost-time accidents and has a record of 445 consecutive days worked with no lost-time accidents.

Mine No. 1 is opened by a drift and is working the Blossburg coal bed, which averages 6 feet in thickness. The production is largely from pillar extraction with exceptionally bad roof conditions. The mine is extremely dry, being in a semi-arid country and at an elevation of 6,500 feet. It is thoroughly rock-dusted, and face regions and broken coal are thoroughly wetted. The mine is rated as gassy. Permissible explosives are used, and shots are fired from outside the mine. Coal is undercut, and loading is both mechanical and hand.

6. Orenda Mine

The Orenda mine of the Davis Coal & Coke Co., Boswell, Pa., operated without a lost-time accident from December 14, 1931 to January 31, 1933, a period of 13-1/2 months, with an average of 180 men working 279,393 man-hours and producing 129,841 tons of coal.

The Orenda mine is opened by a slope into the C Prime coal bed, which averages 60 inches in thickness and has an average pitch upward of 10 percent. About 60 percent of the production was from pillar workings.

7. Steubenville Mine

The Steubenville Mine of the Consumers Mining Co., Steubenville, Ohio, operated without a lost-time accident during 744 days from January 4, 1931, to January 9, 1933, with an average of 34 men working 230,732 man-hours and producing 128,600 tons of coal.

The Steubenville mine is operated through a shaft into the Freeport No. 6 coal bed, which averages 3-1/2 feet in thickness. About 85 percent of the production was from pillar workings.

8. Hull Mine

The Hull mine of the DeBardleben Coal Corporation, Dora, Ala., operated 11 months and 20 days in 1930 without a lost-time accident, with an average of 16 men working 301,763 man-hours and producing 100,699 tons of coal; this mine also operated the first 11 months of 1929 with only 2 lost-time accidents and 7 days lost.

The Hull mine is opened by a drift into the Mary Lee coal bed, which averages about 6-1/2 feet in thickness. The room-and-pillar method of mining is used and pillars are recovered on the retreat. The mine is rated as non-gassy.

9. Weyanoke Mine

The Weyanoke mine of the Weyanoke Coal & Coke Co., Lowe, W. Va., operated without a lost-time accident during 1932 with an average of 100 men working 146,589 man-hours in the production of 125,000 tons of coal.

The Weyanoke mine is opened by a drift into the Pocahontas No. 3 coal bed, which averages 56 inches in thickness.

10. Mossboro Mine

The Mossboro mine of the Black Diamond Coal Mining Co., Mossboro, Ala., operated without a lost-time accident from October 1, 1931 to December 31, 1932, with 144,925 man-hours of exposure and a production of 44,436 tons of coal.

The Mossboro mine is operated through a slope in the Buck coal bed, which averages 40 inches in thickness and dips 20 to 25°. The mine is rated as gassy, and rock-dusting and sprinkling are employed.

11. Block No. 1 Mine

The No. 1 mine of the Block Coal & Coke Co., Block, Tenn., operated without a lost-time accident from May 26, 1931 to July 1, 1933, employing an average of 115 men in the production of 67,832 tons of coal.

The Block No. 1 mine is opened by a drift into the Dean coal bed, which averages 5 feet in thickness.

12. Rockvale No. 3 Mine

The Rockvale No. 3 mine of the Colorado Fuel & Iron Co., Canon City, Colo., operated without a lost-time accident from June 22, 1931 to June 24, 1932, with a production of 41,134 tons of coal. This mine operated from June 1904 to December 31, 1932, or 28-1/2 years, without a fatality.

The Rockvale No. 3 mine is operating through a slope in the Canon coal bed, which is about 6 feet thick.

Table 2. - Records of some mines operated without lost-time accidents

Mine	Operator	Place	Period	Men employed	Coal mined, tons	Exposure, man-hours	Remarks
Nellis	American Rolling Mill co.	Nellis, W. Va.	43 mos.	--	1,080,733	144,831	Applies only to haulage employees. No lost-time accidents.
Hammond	Philadelphia & Reading Coal & Iron Co.	Girardville, Pa.	77 days	508	158,180	312,925	Gassy anthracite mine on pitches up to 65°.
No. 9	Rockhill Coal & Iron Co.	Wood, Pa.	329 days	251	154,000	416,000	Anthracite mine, 10 to 70° pitch. No lost-time accident.
Coke Works	Stonega Coke & Coal Co.	Stonega, Va.	37 mos.	--	Coke		One lost-time accident.
Beech Bottom	Windsor Power House Coal Co.	Windsor Heights, W. Va.	10 mos.	403	203,711 398,827	539,223 --	No fatalities for 14 years. No lost-time accidents.
Brookside	Philadelphia & Reading Coal & Iron Co.	Tower City, Pa.	3 mos.	1,000	144,451	--	5 lost-time accidents.
Nellis	American Rolling Mill Co.	Nellis, W. Va.	7 mos.	300	176,750	--	No lost-time accidents.
Empire	DeBardeleben Coal Corporation.	Empire, Ala.	18 mos.	135	--	--	135 negro miners. No lost-time accidents.
No. 2	Utah Fuel Co.	Castle Gate, Utah	74 days	--	--	--	No lost-time accidents.
No. 3	do.	do.	200 days	30	--	--	do.
Benoit	Black Diamond Coal Mining Co.	Benoit, Ala.	14-1/2 mos.	100	74,987	235,770	1 lost-time accident.

Other Excellent Records

Table 2 shows the records of some mines that have operated for long periods with only few lost-time accidents. It is observed that the haulage employees of the American Rolling Mill Co. mine at Bellis, W. Va., operated for 43 months or 144,831 man-hours, handling 1,080,733 tons of coal without a lost-time accident. The employees of the coke works of the Stonega Coke & Coal Co., Stonega, Va., worked 37 months with 539,227 man-hours of exposure with only one lost-time accident. The records of the two anthracite mines shown in table 2 are remarkable in that the natural conditions present many more hazards than ordinarily found in bituminous mining.

A significant fact noted from the list of mines that have gone for long periods without lost-time accidents is that 35 percent of them are operated by companies that are subsidiaries of companies in other industries. This fact would indicate that essentially the same safety policies as are used in the other industries are effective when applied to coal mines.

METHODS USED TO OBTAIN NO-ACCIDENT RECORDS

It is of interest to know how the good safety records of these mines were attained. Virtually all these records were made possible only by acceptance of responsibility for accidents and accident occurrence by company executives, in the belief that accidents can be prevented. Some of the factors that entered into making these records possible are:

1. In most cases the executives of the company took a personal part in securing the sustained interest of all employees in safety measures, as they realized that safety in and around mines is largely a problem of mine management and mine safety organization.

2. Some type of safety organization was generally established, and all supervising officials were expected to take an active part. As a rule meetings were held monthly, and in at least some cases all employees were required to attend. Accidents occurring at the mine were discussed at these regular meetings, and employees were encouraged to report hazards and dangerous practices observed by them. The safety education of the employees was accomplished largely through these meetings.

3. Safety rules and suggestions were transmitted to the employees through daily verbal instruction by the official.

4. Well-designed bulletin boards were in many instances placed at strategic locations for the instruction of employees.

5. Employees of several mining plants were given a rigid physical examination. New employees were examined and placed in positions for which they were best fitted, and prospective employees who could not meet the rigid requirements were not hired.

6. Rigid discipline was inaugurated and maintained. Carelessness by workmen or officials was not tolerated. Employees who persisted in using careless methods of working were usually discharged, while somewhat less severe penalties were applied to employees whose violation of safety rules or regulations appeared to be due to lack of information rather than to premeditation.

7. Usually wage scales were maintained at a level slightly above the prevalent scale of the district, and employees were given other evidences that the employer was attempting to "play fair" with them.

8. Friendly rivalry among employees was encouraged in some mines by the free distribution of small prizes to those on any section which had worked an entire month without a lost-time accident.

9. Safety was made an important factor in the scheme of production in essentially every mine which had these good safety records.

10. The safety engineer was an integral part of the management in at least some of the mines listed as having achieved eminence in safety performance.

11. All hazards were recognized as potential causes of accidents by several of these mines.

12. A definite program was placed in effect looking to the elimination of such hazards as could be removed from these mines; in other words, the good safety record was brought about by concerted, well-directed effort.

13. Printed rules were posted or issued in pamphlet form governing procedure in dealing with such hazards as could be eliminated; in this manner the employer acquainted the worker not only with the attitude of the company toward safety but also with the minimum requirements of the company as to safe practices.

14. Discipline was maintained from the top of the organization down; close study of the conditions under which these very fine coal-mine accident records were attained appears to indicate that the determining factor was firm but just discipline by a supervisory force able to secure cooperation of the worker.

15. A state of "safety-mindedness" or the appreciation of the disciplinary measures invoked was developed, and in this education not only of the worker but also of the supervisory official was essential.

16. The degree of cooperation of the employees was in large measure secured by the evidence of earnestness with which the company conducted the safety program; where the major responsibility for accident prevention was accepted by the operating officials and definite efforts made to remedy

substandard conditions and eliminate dangerous practices, the worker was usually inclined to try to do his share when told what to do. Some of the improvements made by companies which had these very good safety records were as follows, in general the improvements having been made before the company tried to put the safety campaign before the workers:

- a. The best and safest type of equipment was purchased and put into service.
- b. Moving parts of machinery where feasible were protected by guards.
- c. Haulageways were cleared of debris and tracks overhauled and improved.
- d. Adequate clearance was provided along haulageways.
- e. Loose roof was taken down or properly timbered, in some cases with steel or steel and brick sets.
- f. Improved ventilation was provided and maintained.
- g. Improvements of various kinds were made in blasting practices.
- h. Power wires and equipment, electrical and otherwise, were efficiently installed, guarded, and kept in good repair.

17. Ten of the twelve mines listed in table 1 as having achieved outstanding, no-lost time accident records, had received Bureau of Mines certificates indicating that the entire personnel had been give the Bureau of Mines course in first aid; the training was done before the record was made in most instances and during the making of the record in others.

CONCLUSIONS

On the basis of the 1930 fatal and nonfatal injury record for the bituminous industry of the United States the mines listed in table 1 avoided approximately 6 fatalities and 258 nonfatal injuries during the period of their operation without lost-time accidents. In considering the economic saving to the mining companies by the elimination of these fatalities and injuries it is estimated that a direct saving of approximately \$50,000 or a total saving (directly and indirectly) of about \$300,000 was made; and this does not take into consideration the savings of numerous kinds--economic, humanitarian, social, and otherwise--to the workers whose lives and limbs were kept from jeopardy.

As a rule, a safe mine is an efficient mine, and safety is now fairly well associated with lower rather than higher cost of production. It is significant that approximately 58 percent of the companies shown in table 1 as having operated 1 year or more without a lost-time accident are subsidiary companies, which would indicate that the safety policy of the parent company,

when applied intelligently to its coal mines, produces safety results comparable with those achieved by the parent company in its other industrial operations.

That accidents in coal mines can be very greatly reduced if not entirely eliminated has been conclusively demonstrated by the records of the companies described above. These records were not attained by chance; they were the results of intelligent and sincere efforts on the part of the mine managements who consider the problem of preventing accidents as an important part of coal production, and of the workers whose active cooperation was secured generally by this well-formulated, well-directed educational effort.

UNITED STATES BUREAU OF MINES
SCOTT TURNER, DIRECTOR

INFORMATION CIRCULAR

ACCIDENT EXPERIENCE AND COST IN VIRGINIA COAL MINES
1929 TO 1933, INCLUSIVE



BY

JOSEPH F. DAVIES AND H. B. HUMPHREY

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INFORMATION CIRCULAR

UNITED STATES BUREAU OF MINES

ACCIDENT EXPERIENCE AND COST IN VIRGINIA COAL MINES
1929 TO 1933, INCLUSIVE ¹

By Joseph F. Davies² and H. B. Humphrey³

NATURE OF THIS REPORT

Along with the humanitarian need for preventing accidents the cost of injuries must be considered. As a means of bringing the facts concerning the prevalence of accidents in mining and their direct cost before the operating officials and others interested in accident prevention in the mining industry, the United States Bureau of Mines has recently made studies and published reports dealing with these subjects in several States⁴. This report gives available data on the causes and direct costs of accidents in Virginia coal mines for the 4-year period 1929 to 1932, inclusive.

Mounting compensation costs and insurance rates resulting from mine injuries indicate the necessity of measures to decrease the number and severity of accidents. These direct costs, as well as additional losses of an indirect nature, can be eliminated from the cost of production to the extent that accidents are prevented or reduced in severity.

WORKMEN'S COMPENSATION LAW OF VIRGINIA

The Virginia compensation act was passed in 1918 and amended in 1920, 1922, 1923, 1924, 1926, 1928, 1930, and 1932. It was modeled after the Indiana act; the Georgia and North Carolina acts are very similar. Employers of more than 10 persons are subject to its provisions. Exemption may be claimed by giving formal notice and accepting the legal penalties involved. Farm laborers, domestic servants, and interstate transportation companies are not covered.

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- 1 The Bureau of Mines will welcome reprinting of this paper, provided the following footnote acknowledgment is used: "Reprinted from U. S. Bureau of Mines Information Circular 6763".
 - 2 District engineer, U. S. Bureau of Mines, Norton, Va.
 - 3 Assistant mining engineer, U. S. Bureau of Mines, Norton, Va.
 - 4 See list in appendix, page 13.

The law provides for the payment of compensation on the following schedule:

Death.- If the death results from the accident within 6 years, a weekly payment of 55 percent of the weekly wages, but not more than \$14 or less than \$6 a week for 300 weeks, but not over \$5,000 and burial expenses not exceeding \$150.

The above is to be paid to a total dependent, but if dependence is only partial compensation will be in the corresponding ratio. If no dependents are left residing in the United States or Canada compensation is limited to \$1,000.

Total incapacity.- During such period as total incapacity exists, a weekly compensation equal to 55 percent of the average weekly wages of the injured, but not more than \$14 or less than \$6 per week, and in no case for more than 500 weeks or over \$5,600.

Partial incapacity.- When the incapacity is partial a weekly compensation equal to 55 percent of the difference between the average weekly wages of the injured before the injury and those he is able to earn thereafter, but not more than \$14 a week or for more than 300 weeks.

Schedule of injuries, rate, and period of compensation.- When the following injuries occur, the incapacity is deemed to continue for the number of weeks specified, and the compensation is 55 percent of the average weekly earnings.

Loss of	Period, weeks	Loss of	Period, weeks
Thumb	60	Hand	150
Index finger	35	Arm	200
Second finger	30	Foot	125
Third finger	20	Leg	175
Fourth finger	15	Eye	100
Big toe	30	Hearing of 1 ear	50
Other toe	10	Disfigurement	50
First joint, only $\frac{1}{2}$ of above			

Loss of both hands, feet, or eyes or combination of any two is equivalent to total incapacity.

Waiting period.- No compensation is allowed for the first 7 calendar days of incapacity resulting from an injury, but compensation begins with the eighth day, unless such incapacity continues for more than 6 weeks, in which case it is allowed from the first day of incapacity. Before July 1930 the waiting period was 10 days.

Medical service.— The employer must furnish such medical attention as the case requires, for not exceeding 60 days after an accident. The Industrial Commission may extend this period in particular cases, but not in excess of 180 days.

INDUSTRIAL COMMISSION AND STATISTICAL DEPARTMENT

The law is administered by the Virginia Industrial Commission, composed of three members appointed by the Governor. Under it a statistical department has been formed, with two main purposes. The first is presentation of information to employers and others interested in the field of accident prevention; the second is determination of the loss costs under the compensation law, for use in rate-making procedure. In addition, the department collects data necessary for administrative use. Reports sent out to coal-mining officials show essentially the accident situation in their companies and in relation to others in the field. Companies are designated by code letters, each company knowing its own code letter only⁵. This type of service by the statistical department has been instrumental in great improvement in the accident records of certain companies. This cooperation between State departments and corporations has been found effective in essentially every instance where it has been practiced. The statistical department receives reports from every insurance company writing compensation insurance in Virginia on every risk carried and on every injury case. These are checked with the records of the Industrial Commission and basic figures assembled for rate-making procedure.

ACKNOWLEDGMENTS

The cooperation of the Industrial Commission, particularly that of Frank P. Evans in supplying the necessary data, made this report possible. Figures on employment and certain other items were supplements by data furnished by W. W. Adams, chief statistician of the Demographical Division, United States Bureau of Mines.

ACCIDENT RECORD

For the period covered by this report (1929 to 1932) Virginia ranked eighth in the amount of coal produced each year among 24 coal-mining States. The standing of Virginia in relation to the others according to its accident record is shown in table 1. The close relation between the percentages of total production and fatalities given in this table show that normally the fatalities in the coal mines of Virginia are average in number for the amount of coal produced. Had there not been 54 deaths from explosions in Virginia the fatality record for the State in 1932 would have been much better than the average of the United States. The standing of each of these 24 States, according to fatal and nonfatal accident records for 1931, is given in table 2. For this year Virginia occupies a position slightly better than the United

⁵ See Appendix, page 15.

States average in both fatal and nonfatal rates. Virginia's production is slightly less than that of Alabama and a little more than that of Colorado. Both of these States have a better fatality record than Virginia; and, although Colorado's injury rate is somewhat greater than that of Virginia, Alabama's is very much lower. No close comparisons can be drawn between any particular States on the basis of accident rates alone. Hazards may differ greatly so that a good record in one case may reflect naturally advantageous conditions and in another effective accident-prevention methods. However, where wide differences in rates appear between States producing over a million tons the reasons should be sought and corrective measures taken to save lives and reduce mining costs. Among the 16 States having a record better than the average for the United States as a whole, Alaska, Michigan, and Texas each produced less than a million tons.

Table 1. - Fatal mine-accident record of Virginia compared with that of the 24-coal mining States ^{1/}

Year	No. of fatalities		Percentage of--		States having greater no. of fatalities	Deaths per million				States having lower frequency rate
			Total produc- tion	Total fatali- ties		Man-hours		Tons		
	Va.	U. S.				Va.	U. S.	Va.	U. S.	
1929	43	2,187	2.1	2.0	8	1.78	1.37	3.37	3.59	12
1930	46	2,053	2.0	2.2	8	2.43	2.06	4.22	3.84	17
1931	31	1,463	2.2	2.1	6	1.81	1.82	3.19	3.31	15
1932 ^{2/}	71	1,168	2.2	6.1	4	5.70	1.80 ^{3/}	9.20	3.29	22

^{1/} Bulls. 355 and 373; 1932 figures from preliminary reports of Demographical Division, U. S. Bureau of Mines.

^{2/} Includes 54 fatalities from gas and dust explosions.

^{3/} These rates are tentative.

Table 3 includes the record of mines of the companies reporting to the Industrial Commission, and the figures may differ slightly from similar data published by the United States Bureau of Mines. Figures given by the Industrial Commission include more than 98 percent of the tonnage mined in the State. The number of accidents shown differs from that given by the Bureau, since some are included by the Commission that were compensable but not regarded by the Bureau as chargeable to the industry.

The depression is evident in the decreased number of mines, tonnage produced, operating time, and men employed. Consistent reduction in accidents would have given the State a good record in 1932, had there not been 54 deaths from mine explosions. This improvement was accomplished by real co-operative effort of the officials and men of several leading companies, and State and national agencies interested in accident prevention in coal mines. The greatest factor in bringing about the improvement shown in table 3 was the very noteworthy progress made by three large companies. Their record is given in table 4 and that of the others in table 5.

Table 2. - Relative standing of States in 1931,
according to fatality and injury rates
per million man-hours worked
in coal mines 1/

State	Fatality rate	State	Injury rate
Michigan	-	Alaska	28.19
Alaska	-	Arkansas	51.40
Texas	0.73	Tennessee	56.99
Missouri96	North Dakota	57.64
Alabama	1.00	Alabama	62.10
Iowa	1.11	Kansas	64.72
Maryland	1.19	New Mexico	64.94
Kansas	1.33	Missouri	68.00
New Mexico	1.38	Kentucky	72.53
Tennessee	1.56	Maryland	78.16
Pennsylvania	1.60	Montana	79.28
Illinois	1.68	Ohio	81.11
Colorado	1.71	Virginia	81.84
Ohio	1.74	West Virginia	82.03
Kentucky	1.76	Michigan	85.80
Virginia	1.81	Colorado	95.41
United States average ..	1.82	United States average ..	99.89
Montana	1.90	Iowa	100.00
Arkansas	2.23	Pennsylvania	110.27
Oklahoma	2.33	Washington	112.74
Washington	2.38	Oklahoma	120.29
West Virginia	2.53	Wyoming	123.03
Wyoming	2.56	Indiana	129.37
Utah	3.10	Texas	140.81
North Dakota	3.26	Illinois	151.93
Indiana	3.71	Utah	153.10
Other States	10.86	Other States	14.49
United States bituminous	1.81	United States bituminous	90.57
Pennsylvania:		Pennsylvania:	
Bituminous	1.29	Bituminous	89.49
Anthracite	1.84	Anthracite	126.55

1/ From Bureau of Mines Bull. 373, Coal-Mine Accidents in the United States, 1931.

Table 3. - Relation of accidents to operation in Virginia coal mines,
1929 - 1932

Year	Mines oper- ating	Tons produced	Average days active	No. employed	Accidents			
					Fatal	Nonfatal compensable	Non- compensable	Total
1929	89	12,310,248	249	12,835	57	1,414	1,000	2,471
1930	87	10,884,613	200	12,397	47	1,200	752	1,999
1931	83	9,601,843	185	11,613	32	1,151	430	1,613
1932	77	7,307,172	144	10,143	75	793	235	1,103
Totals	336	40,103,876	- -	46,988	211	4,558	2,417	7,186
Average	84	10,025,777	195	11,747	53	1,140	604	1,800

Table 4. - Accident record of three large companies showing greatest
improvement in accident reduction

Year	No. employed, percent of Va. total	Tons produced, percent of Va. total	Lost-time injuries		Accidents per million-	
			Number	Percent of Va. total	Tons	Man-hours
1929	45	47	1,005	40.7	172	86
1930	44	46	664	33.2	131	75
1931	43	45	444	27.6	103	59
1932	39	44	203	18.3	63	44
Total or average	43	45	2,316	32.2	127	71

Table 5. - Accident record of 40 other companies

Year	No. employed, percent of Va. total	Tons produced, percent of Va. total	Lost-time injuries		Accidents per million-	
			Number	Percent of Va. total	Tons	Man-hours
1929	55	53	1,465	59.3	225	105
1930	56	54	1,335	66.8	225	119
1931	57	55	1,169	72.4	221	118
1932	61	56	900	43.7	220	123
Total or average	57	55	4,869	67.8	223	114

The latest available figures for the United States as a whole are given in Bureau of Mines Bulletin 373 for 1931. Based on these data, there were 1.85 lost-time injuries per million man-hours worked. In 1932 the three companies grouped in table 4 had reduced the number of their lost-time injuries per million tons to a third of those experienced in 1929 and on a man-hour basis had cut the rate in half. The remaining companies, representing approximately 55 percent of the production, showed no improvement during the period, although they include in their number a few smaller operations having good records.

It may be noted that the actual records of both groups on a man-hour basis are probably better than shown, as they were calculated on a nominally reported working shift of 8.1 hours, whereas the actual working shift was probably much longer.

Table 6. - Causes of fatal accidents in Virginia coal mines, 1929 to 1932

Cause	Number of fatalities					Rate per million-	
	1929	1930	1931	1932	Total	Tons	Man-hours
Falls of coal or slate	36	24	21	12	93	2.32	1.23
Haulage	13	19	7	5	44	1.10	.58
Explosives and explosions	4	1	1	55	61	1.52	.81
Electricity	1	0	0	2	3	.07	.04
All other	3	3	3	1	10	.25	.13
All causes	57	47	32	75	211	5.26	2.79

Table 7. - Causes of nonfatal accidents in Virginia coal mines, 1929 to 1932

Cause	Number of accidents					Rate per million-	
	1929	1930	1931	1932	Total	Tons	Man-hours
Falls of coal or slate	383	660	550	437	2,530	63.1	33.4
Haulage	550	510	400	188	1,648	41.1	21.7
Explosives and explosions	37	25	10	11	83	2.1	1.1
Handling objects and tools	591	442	395	227	1,655	41.2	21.8
Falls of persons	84	55	50	36	225	5.6	3.0
Electrical	21	18	11	7	57	1.4	0.8
Mechanical	58	70	59	21	208	5.2	2.8
Miscellaneous	190	172	106	101	569	14.2	7.5
All causes	2,414	1,952	1,581	1,028	6,975	173.9	92.1

1/ 1930 and 1931 segregations completed from Bureau of Mines Bulletins 355 and 373.

CAUSES OF ACCIDENTS

The data in table 6 are taken from the reports of the industrial commission. Explosions and explosives accidents are considered in one group, as in the State industrial accident records.

The rate for the 4-year period 1929 to 1932 (1.23 fatalities per million man-hours from falls of roof and coal) and the nonfatal rate (33.4 injuries per million man-hours) are very close to the 1931 average of 1.28 and 33.2, respectively. Roof and coal falls caused 44 percent of the Virginia fatalities from 1929 to 1932, with approximately 27 nonfatal injuries to each fatality from this cause.

Fatalities from haulage accidents were 0.58 per million man-hours and nonfatal injuries 21.7; in 1931, the rates for the United States as a whole were 0.35 and 23.04. Haulage accidents caused nearly 21 percent of the fatalities for the 4-year period in Virginia, and for each fatal accident there were approximately 38 nonfatal injuries. Because of the 54 deaths from explosions in 1932 the rate from that cause is high. For the 4 years the ratio of fatal to nonfatal injuries from explosives and explosions was 1 to 1.4. Accidents from electricity were few; the fatal and nonfatal rates for the State over the 4-year period were about half the average for the country as a whole. For the 4-year period the ratio of fatal to nonfatal injuries in Virginia was 1 to 33; for the 3 years 1929 to 1931, excluding the heavy fatality record of 1932, the ratio was 1 to 44, which is probably a normal ratio for the State. The ratio for the United States for the 2 years 1930 and 1931 was 1 fatality to 52 nonfatal injuries.

ACCIDENT SEVERITY AND COSTS

Accident costs and available information on days lost are shown in table 3. The time lost on the national scale for permanent partial disabilities could not be shown because detailed information as to the exact nature of the disability was not available. The average time lost from compensable temporary injuries was 48.3 days and from noncompensable lost-time injuries 4.1 days. The average for all temporary lost-time injuries was 30.9 days. Apparently the average time lost per temporary injury varies slightly from year to year, with no particular trend. The change from 5 to 3 days lost from noncompensable injuries is based on the change from a 10-day to a 7-day waiting period in June 1930.

The cost of compensation per fatal accident in any year varies according to the average number of dependents and has ranged from \$2,400.30 in 1929 to \$2,988.97 in 1931. Compensation for permanent partial disabilities averaged \$498.18 per case, decreasing from \$534.65 in 1929 to \$421.60 in 1932. This cost for 1932 may be increased by additional awards to open cases on future developments. The cost of compensation for temporary disabilities has not varied much from the average of \$46.11 in any year. The average compensation per nonfatal compensable injury was \$88.26.

Table 8. - Severity and cost of accidents, 1929-32

Description	1929	1930	1931	1932	1929-32
Fatal claims:					
Number	57	47	32	75	211
Cost:					
Compensation ..	\$141,828.00	\$135,291.00	\$90,847.00	\$185,932.00	\$554,498.00
Funeral & medical, est.	8,550.00	7,050.00	4,800.00	11,250.00	31,650.00
Total direct. .	150,378.00	142,941.00	95,647.00	197,182.00	586,148.00
Ave. compensation cost. ...	2,400.30	2,891.30	2,933.97	2,629.09	2,627.95
Ave. medical cost, est. ...	150.00	150.00	150.00	150.00	150.00
Ave. total direct cost	2,550.30	3,041.30	3,133.97	2,779.09	2,777.95
Permanent total disability claims:					
Number	1	1	1	1	4
Cost:					
Compensation ¹ ..	4,500.00	4,250.00	4,500.00	4,500.00	17,750.00
Medical ¹	250.00	250.00	250.00	250.00	1,000.00
Total direct ..	4,750.00	4,500.00	4,750.00	4,750.00	18,750.00
Permanent partial disability claims:					
Number	244	222	233	159	858
Days lost (actual lost time plus State award) ...	85,901	71,158	72,291	51,253	280,603
Ave. per case .	352	321	310	322	327
Cost:					
Compensation ¹ ..	\$130,453.00	\$114,562.00	\$115,394.00	\$67,033.00	\$427,442.00
Medical, estimated	24,400.00	22,200.00	23,300.00	15,900.00	85,800.00
Total direct. .	154,853.00	136,762.00	138,694.00	82,933.00	513,242.00
Ave. compensation cost	534.65	516.05	495.25	421.60	498.18
Ave. medical cost, est. ...	100.00	100.00	100.00	100.00	100.00
Ave. total direct cost ..	634.65	616.05	595.25	521.60	598.18

¹/ Open cases are included. In a few cases the cost over present award is estimated.

Table 8. - Severity and cost of accidents, 1929-32--Continued.

Description	1929	1930	1931	1932	1929-32
Temporary disability claims:					
Number	1,169	977	917	6331	3,696
Days lost	49,951	54,790	44,783	29,118	178,642
Ave. per case	41.9	56.2	48.8	46.0	48.3
Costs:					
Compensation	\$53,455.00	\$50,326.00	\$39,848.00	\$26,778.00	\$170,407.00
Medical (est.) ...	46,760.00	43,965.00	36,680.00	22,155.00	149,560.00
Total direct costs	100,215.00	94,291.00	76,528.00	48,933.00	319,967.00
Ave. compensation costs	45.73	50.49	43.45	42.30	46.11
Ave. medical cost, (est.)	40.00	45.00	40.00	35.00	40.46
Ave. total direct cost	85.73	95.49	83.45	77.30	86.57
Noncompensable temporary cases:					
Number	1,000	752	430	235	2,417
Days lost	5,042	3,008	1,300	710	10,060
Ave. per case	5.0	4.0	3.0	3.0	4.1
Costs:					
Medical (est.) ...	\$8,000.00	\$6,016.00	\$3,225.00	\$1,762.00	\$19,003.00
Ave. medical cost (est.)	8.00	8.00	7.50	7.50	7.87
Total nonfatal injuries:					
Number	2,414	1,952	1,581	1,028	6,975
Cost:					
Compensation	\$188,408.00	\$169,138.00	\$159,742.00	\$98,311.00	\$615,599.00
Medical (est.) ...	79,410.00	72,431.00	63,455.00	40,067.00	255,363.00
Total direct	267,818.00	241,569.00	223,197.00	138,378.00	870,952.00
Ave. compensation.	78.04	86.65	101.04	95.63	88.26
Ave. medical	32.90	37.10	40.14	38.98	36.61
Ave. total direct.	110.94	123.75	141.18	134.61	124.87

Table 8. - Severity and cost of accidents, 1929-32--Concluded.

Description	1929	1930	1931	1932	1929-32
Fatal plus nonfatal:					
Number	2,471	1,999	1,613	1,103	7,186
Cost:					
Compensation ^{1/}	\$330,236.00	\$305,029.00	\$250,589.00	\$284,243.00	\$1,170,097.00
Funeral and medical (est.)	87,960.00	79,481.00	68,255.00	51,317.00	287,013.00
Total direct ..	418,196.00	384,510.00	318,844.00	335,560.00	1,457,110.00
Ave. compensation cost	133.64	152.59	155.36	257.70	162.83
Ave. medical cost	35.60	39.76	42.31	46.52	39.94
Ave. total direct cost ..	169.24	192.35	197.67	304.22	202.77

^{1/} Open cases are included. In a few cases the cost over present award is estimated.

Total compensation costs for fatal accidents averaged \$138,625 and for non-fatal accidents \$153,900, making an average yearly total of \$292,525. Medical costs were estimated from the available data, since several companies provide such service themselves and no figures were given. These estimated costs agree closely with those given in some similar reports on other States. (See Appendix.) The average total funeral and medical cost of all fatal and lost-time injuries per year was \$71,753, or \$39.94 per case. The combined total direct costs averaged \$364,278 a year, or \$202.77 per lost-time injury, including fatalities.

DIRECT AND INDIRECT COSTS OF ACCIDENTS

Direct accident costs considered in this report include medical, hospital, funeral, and compensation charges. The indirect costs are more intangible, but usually include the following items:

1. Time lost by injured employee.
2. Time lost by other affected employees.
3. Injury to property.
4. Loss of production.
5. Continued overhead.

The estimated costs of the indirect losses to the employer due to an accident are approximately four times the direct costs, according to H. W. Heinrich of the Travelers Insurance Co. On this basis the average indirect cost of accidents to the operator in Virginia coal mines is \$1,457,110 per year and the total cost \$1,821,383.

CONCLUSION

The average direct cost of accidents in Virginia coal mines is approximately 3.6 cents per ton, and the total cost therefore would be about 18 cents. These costs are high enough to indicate plainly why accident prevention should be fostered and studied by every group connected with the industry, aside from the humanitarian obligation. This cost must come out of profit, therefore all employees, as well as the management, have a vital interest in accident reduction. Considerable reduction in accident cost and frequency can be made with little or no expense other than sincere cooperative efforts by the management and employees.

A P P E N D I X

The following publications by the United States Bureau of Mines deal with the subject of accident costs:

The Cost of Accidents to Industry, by F. S. Crawford.
Inf. Circ. 6333, 1930, 10 pp.

Accident Experience and Cost of Accidents in Washington Coal Mines, by S. H. Ash. Inf. Circ. 6529, 1931, 18 pp.

Accident Experience of the Coal Mines of Utah for the Period 1918 to 1929, by A. L. Murray and D. Harrington. Inf. Circ. 6530, 1931, 26 pp.

Accident Experience and Cost in Pennsylvania Anthracite and Bituminous Mines, 1926-30, by W. J. Fene. Inf. Circ. 6618, 1932, 29 pp.

Accident Experience and Cost in Tennessee Coal Mines, by F. E. Cash. Inf. Circ. 6664, 1932, 8 pp.

Ten Years of Fatal Accidents and Two Years of Accident Costs in Indiana Coal Mining, by C. A. Herbert. Inf. Circ. 6672, 1932, 12 pp.

Accident Experience and Costs in Colorado Metal Mines, by E. H. Denny and E. A. Amundsen. Inf. Circ. 6713, 1933, 23 pp.

Accident Experience and Cost of Accidents at Washington Metal Mines and Quarries, by S. H. Ash. Tech. Paper 514, 1932, 35 pp.

C O P Y

(State Seal)

Commonwealth of Virginia
 Department of Workmen's Compensation

Statistical Dept.
 Frank P. Evans,
 Statistician.

INDUSTRIAL COMMISSION OF VIRGINIA

Richmond

August 1933

C O A L , M I N E S

Accident frequency to tons of coal produced

January 1 to June 30, 1933

Average No. employees	Tons of coal produced	*Lost-time injuries	Jan.-June 1933	Jan.-June 1932
10,122	3,824,838	425	8,999	6,426

* Industrial accidents which caused injuries resulting in loss of time
 from work of one day or more.

Fatal cases January 1 to June 30, 1933 5
 Fatal cases January 1 to June 30, 1932 63

Average tons produced per fatal case, Jan. 1 to June 30, 1933 - 764,967

(Type of bulletins sent to operators by Statistical Dept.)

FMS.

August 1933

Tons produced per lost-time injury

Company	January 1 to June 30, 1933	1932
"A"	8,276	7,115
"B"	15,766	11,293
"C"	16,249	2,167
"D"	1,845	2,633
"E"	12,604	12,781
"F"	2,727	* 3,437
"G"	6,264	5,054
"H"	3,493	-
"I"	7,640	2,740
"J"	17,433	3,756
"K"	1,807	4,210
"L"	2,461	2,386
"M"	7,816	4,158
"N"	7,133	3,265
"O"	7,284	4,488
"P"	54,364	25,062
"Q"	3,563	8,727
"R"	6,765	5,004
"S"	5,517	2,833

Your code designation is

* Tons produced without a lost-time injury.

UNITED STATES BUREAU OF MINES
SCOTT TURNER, DIRECTOR

INFORMATION CIRCULAR

EXPLOSIONS IN ILLINOIS COAL MINES

1883 TO 1932



BY

C. A. HERBERT

INFORMATION CIRCULAR

UNITED STATES BUREAU OF MINES

EXPLOSIONS IN ILLINOIS COAL MINES
1885 to 1932¹

By C. A. Herbert²

PURPOSE OF THIS REPORT

The old fatalistic idea that industrial accidents were a necessary part of industry and that little, if anything, could be done about them has been succeeded by realization that a majority of the hazards causing these accidents were man-made and that if they could be recognized most of them could be eliminated and the accidents prevented. Recognition of these facts has brought about substantial reduction of accidents in many coal mines, as well as in other industries.

Knowledge of the hazards and their elimination has been obtained largely through a study of individual accidents and the causes which led up to them. With this thought in mind the available data on coal-mine explosions in Illinois were studied.

EXPLOSIONS IN ILLINOIS COAL MINES

The data for this study were obtained largely from publications of the Illinois Department of Mines and Minerals and from information gathered by the United States Bureau of Mines.

Information is not available on coal-mine explosions in Illinois before 1882, as the first coal report for the State was issued that year. The first explosion of which there is available record occurred in 1884, and the only information given is that 10 men were killed by a gas explosion, presumably due to open lights and lack of ventilation.

From 1885 to 1896, inclusive, there is no record available of occurrence of any mine explosions, although during this period there were 16 fatalities from ignitions of gas. The second explosion on which a record is available occurred in 1897; four men were killed. During the next 3 years the mines apparently were free from either explosions or fatalities due to ignitions of gas or blown-out shots.

¹The Bureau of Mines will welcome reprinting of this paper, provided the following footnote acknowledgement is used: "Reprinted from U. S. Bureau of Mines Information Circular 3734."

²Supervising engineer, U. S. Bureau of Mines, Vincennes, Ind.

From 1901 to January 1, 1933, inclusive, the only years free from explosions were 1906, 1912, and 1925; in each of these years, however, men were killed either by gas ignitions or blown-out shots of black blasting powder.

Table 1 summarizes the data available on explosions in Illinois from 1883 to 1932, in which marked violence occurred. Under the heading "Other Cases" are listed the number of gas ignitions in which men were killed and the number of men killed each year from this cause, as well as the number of minor explosions due to blown-out shots of black blasting powder and the number of men killed each year from this cause.

Fifty-seven explosions of limited or considerable extent, for which more or less complete data are available, killed a total of 427 men and injured 151; 72 gas ignitions killed 88 men; 151 minor explosions due to blown-out shots of black blasting powder killed 179 men - a total of 764 fatalities from 1883 to 1932, inclusive, or an average of about 15 per year.

The number killed in the cases listed, for which some data are given, ranged from 1 to 54 with an average of about 9 per explosion.

Fatality Rates

Table 2 presents the fatality rates for gas and dust explosions per million tons of coal produced and per million man-hours of surface and underground work.

From 1883 to 1900 most of the coal was produced from the comparatively nongassy beds of the northern and central part of the State; and although black blasting powder was used almost entirely as an explosive, it was not employed with the utter disregard for safety that later marked its use in the solid shooting mines of the section. This fact is reflected in the low fatality rates from explosions during this period.

Development of the deeper, more gassy seams of southern Illinois began about 1901; the era of reckless shooting with black blasting powder in solid shooting mines also began about this time and no doubt accounts for the increased fatality rates from explosions after that date.

The years 1929, 1930, and 1931 show very marked improvement in the fatality rates from mine explosions. The record for 1932, however, was the worst in loss of life from explosions in the history of the industry in Illinois, due to a serious disaster in December of that year.

Causes of Explosions in Illinois

Table 3 summarizes the causes of the 57 explosions listed in table 1. It will be observed that open lights caused 27, or 47.5 percent, of the explosions; blown-out shots of black blasting powder caused 12, or 21 percent; mine fires 6, or 10.5 percent; electricity 5, or 8.3 percent; explosives 4, or 7 percent; striking matches 3, or 5.2 percent.

TABLE 1. - Gas and dust explosions in bituminous mines of Illinois, 1851-1932

Year	County	Shaft, nongassy	No. of miners	Time of day	Type	Extent	Limited by	Cause of accumulation	Open lights (r)	Source of ignition	Other factory	Possible means of prevention of time	Other Cases					
													Killed	Injured	Escaped	Gas ignited	Killed	Blown-out shots
1 1851 Jan. 6	Randolph	150	40	40	Gas	-	-	Lack of ventilation	-	Open lights (r)	-	-	10	70	-	-	-	Blown-out shots
1854 1/	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
1855	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
1856	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
1857	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
1858	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
1859	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
1860	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
1861	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
1862	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
1863	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
1864	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
1865	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
1866	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
1867	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
1868	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
1869	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
1870	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
1871	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
1872	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
1873	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
1874	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
1875	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
1876	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
1877	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
1878	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
1879	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
1880	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
1881	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
1882	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
1883	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
1884	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
1885	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
1886	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
1887	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
1888	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
1889	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
1890	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
1891	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
1892	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
1893	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
1894	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
1895	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
1896	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
1897	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
1898	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
1899	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
1900	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
1901	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
1902	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
1903	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
1904	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
1905	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
1906	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
1907	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
1908	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
1909	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
1910	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
1911	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
1912	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
1913	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
1914	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
1915	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
1916	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
1917	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
1918	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
1919	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
1920	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
1921	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
1922	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
1923	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
1924	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
1925	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
1926	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
1927	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
1928	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
1929	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
1930	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
1931	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
1932	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-

1/ Years marked with an asterisk are those in which there were no major accidents, or in which no accidents occurred.

TABLE 1. - Gas and dust explosions in bituminous mines of Illinois, 1885-1932--Continued

Case No.	Date	County	Shaft, gas or nongassy	Time of day	Average employment	Men in mine	Type	Extent	Limited by	Cause of accumulation	Source of ignition	Explosion				Other cases			
												Killed	Injured	Rescued	Uninjured	Gas ignitions	Killed	Blown out shoots	Total fatalities
21	1914: Aug. 23	Franklin	Shaft, gassy		1,500	300	Gas	Limited	Amount of gas	Lack of ventilation	Open lights	Failure to test for gas in new rooms	Closed lights, closer supervision and inspection	2	298	2	2	-	-
22	Oct. 1	Perry	Shaft, nongassy		1,500	312	Dust	Wide-spread	-	-	Black powder	Explosives being delivered on entry	Closed lights, closer supervision and inspection	2	-	-	-	-	-
23	Oct. 14	Franklin	Shaft, gassy		2,600	357	Gas and dust	do	Expansion	Deranged ventilation	Open lights	Gas reported, stopping being repeated	Better ventilation, inspection not more than 7 hours before starting time. Closed lights.	52	19	256	-	-	57
24	1915: Apr. 5	Montgomery	Shaft, nongassy		2,300	600	Gas	Limited	Inert material	Inadequate ventilation	do	Occurred as men reached face of entry	do	11	-	289	-	6	9
25	July 27	Franklin	Shaft, gassy		2,677	416	Gas	do	Expansion	do	do	Occurred Monday morning, last examination Saturday night	Closer examination for gas, closed lights, rock-dusting	9	3	399	-	-	29
26	1916: Mar. 16	Perry	Shaft, nongassy		2,000	300	Dust	do	-	-	Ignition of keg of powder	None	Permissible explosives	2	1	297	5	3	3
27	July 30	Christian	do		1,050	315	Gas	Extensive	Amount of gas	Fan shut down	Open lights	Mine not examined before men entered	Closer examination, running fan longer, closed lights	4	2	-	-	-	-
28	Dec. 13	Franklin	Shaft, gassy		4,000	240	Gas	Limited	do	Cave-in, lack of ventilation	do	None	Closed lights, more careful inspection	1	7	242	-	-	15
29	1917: Feb. 23	Sangamon	Shaft, nongassy		1,000	201	Dust	Fairly extensive	Expansion, combustible inert material	Blown-out shot	Blown-out shot	Solid shooting, overcharged shots	Permissible explosives, undercutting	4	-	3	4	4	5
30	Mar. 12	Saline	Shaft, gassy		1,150	180	Gas and dust	Gas and dust	Expansion, inert material	Door left open	Open lights	Fire boxes using open lights	Closed lights	4	-	5	-	-	-
31	June 3	Williamson	Shaft, nongassy		3,000	394	Gas	do	Amount of gas	Removal of curtain	do	Gas accumulated in old workings	Closed lights, better supervision and inspection	9	2	9	-	-	-
32	July 26	Franklin	Shaft, gassy		1,500	275	Gas and dust	do	Amount of gas	Squeeze, lack of ventilation	Trolley locomotive	Gas accumulated in worked-out territory, pushed out by fall	Better supervision and inspection	1	1	274	-	-	-
33	Nov. 29	do	do		3,500	423	Gas	Wide-spread	-	Door left open	Open lights	No examination	Closed lights, rock-dusting, better examination, supervision	17	-	-	-	-	-
34	Dec. 5	do	do		4,000	700	Dust	Limited	do	Sealing mine mine fire	Open lights and fire	4 explosions	Closed lights	3	13	3	-	-	47
35	1918: Feb. 22	Macoupin	Shaft, nongassy		2,200	475	Dust and powder	do	Inert material, expansion	None	Explosives	Explosion of 26 kegs black blasting powder, ignition by electricity	Deliver powder at night with electricity off	4	-	471	-	-	-
36	Feb. 23	Sangamon	do		1,000	163	Dust	Wide-spread	-	-	Blown-out shot, black blasting powder	-	-	4	-	-	5	8	3
37	Aug. 14	Saline	Shaft, gassy		850	200	Gas and dust	Gas and dust	Amount of gas	Lack of ventilation	Open lights	-	-	2	-	198	-	-	-
38	Sept. 28	Franklin	do		3,100	500	Gas	do	Rock-dust	Cut through into old workings	do	Gas ignition outby seals	Closed lights	21	-	13	-	-	42
39	1919: Jan. 6	do	Shaft		3,435	500	Gas	do	do	do	do	Mine manager failed to carry out instructions of State inspector	-	3	1	496	-	3	5
40	May 12	Marion	Shaft, nongassy		1,400	518	Dust	-	-	-	Blown-out shot.	-	Better supervision	4	-	-	-	-	12
41	1920: Nov. 11	Williamson	do		350	112	Dust	Limited	Size of mine	None	Blown-out shot, black blasting powder	None	Permissible explosives	2	-	2	2	5	6
42	Jan. 29	do	do		2,000	350	Gas	do	-	None	do	-	Undercutting, permissible explosives	1	-	3	-	-	11
43	1921: Jan. 14	Franklin	Shaft, gassy		4,000	400	Gas and dust	do	Rock-dust, expansion	Squeeze in old workings	Lighted march from smoking	Squeezing forced gas onto haulage	Closer inspection and supervision	2	7	-	1	1	2
44	Feb. 14	Saline	do		1,240	254	Gas	do	Amount of gas	Lack of ventilation	Open lights	-	Closer supervision, ventilation	3	-	251	-	-	-
45	Feb. 23	Marion	Shaft, nongassy		3,000	450	Dust	do	Expansion, inert material	-	Blown-out shot, black blasting powder	Solid snorting, badly placed, overcharged shots	Undercutting, permissible explosives	3	-	2	-	-	-
46	Aug. 31	Saline	Shaft, gassy		7,000	500	Gas	do	Amount of gas	Poor ventilation	Open lights	Gas in entries not working, new drilled into, gas lighted at hole	Closer inspection and supervision, closed lights	11	-	489	-	-	25

TABLE 1. - Gas and dust explosions in bituminous mines of Illinois, 1871-1912--Continued

Case No.	Date	County	Type of mine or shaft	Persons employed	Average per day	Type and extent of explosion	Cause of explosion	Source of ignition	Other factors	Possible means of prevention at time	Explosion							Other cases				
											Killed	Injured	Escaped uninjured	Gas ignitions	Killed	Black blasting powder	Total yearly fatalities	Killed	Injured	Escaped uninjured	Gas ignitions	Killed
47	1922: Sept. 23	Williamson	Shaft, nongassy	2,500	785	Gas	Limited amount of gas.	Open lights	Entries not working, not examined	Closer inspection and supervision, closed lights, and sealing of unventilated sections	5	-	-	3	3	7	10	18				
48	1923: Dec. 26	Franklin	Shaft, gassy	6,000	850	do	Amount of gas, expansion	do	Gas accumulated in pillar work	Closed lights, better inspection and supervision	2	14	874	10	11	9	10	25				
49	1924: Jan. 25	Williamson	do	3,000	400	Gas and dust	Expansion and non-combustible	do	Recovery of material in worked-out section	Closer supervision and inspection, rock-dust, light	35	7	330	-	-	10	14	47				
50	1925: Jan. 29	Franklin	Shaft, gassy	10,000	1,200	Gas and dust	Limited moisture, Lack of ventilation and rock-dust	Striking match	Fire boss lighted safety lamp	Permissible flame safety lamp, rock-dusting, better ventilation	5	-	1,132	2	2	10	13	20				
51	1927: Mar. 30	Saline	do	1,500	216	do	Moisture, inflammable in dust	Open lights	Moving body of gas, men in mine	Better ventilation, closed lights, permissible machinery, rock-dusting, better supervision	8	3	185	2	2	6	6					
52	Dec. 20	Williamson	Shaft, nongassy	3,400	450	Gas	do	Electricity	Fire boss rode to face on haulage locomotive	Hot allow fire boss to ride locomotive to face, examine mine before shift	7	-	23	-	-	-	23					
53	1928: Jan. 9	Franklin	Shaft, gassy	4,200	750	Gas and dust	Expansion, inflammable in dust	Striking match, smoking	Closed-light mine, examined too long before shift started	Examination not over 2 hours before shift. Search for matches, rock-dusting	21	-	611	2	2	4	5	28				
54	1929: Dec. 1	do	do	4,500	600	do	Expansion, Caving of abandoned rock-dust workings	Electricity	Inadequate inspection of old works close to open equipment	Permissible equipment, better ventilation, inspection, and supervision	7	-	17	-	-	2	3	10				
55	1930: Mar. 18	do	do	6,000	300	do	Door left open	do	Open-type equipment closed lights	Permissible equipment, examine line for gas by machines men	3	3	294	-	-	-	-	3				
56	1931: May 28	Jackson	do	4,900	470	Dust	Rock-dust	do	Dust cloud caused by wrecked trip	Rock-dusting, permissible equipment, recess legs of timber sets in ribs	2	1	80	1	1	1	2	5				
57	1932: Dec. 24	Shelby	Shaft, nongassy	400	104	Gas and sive dust	Amount of poorly built seals to gas and old workings and sudden drop of tide in barometer, poor road ventilation	Open lights	Better ventilation, better sealing of old "works", closed lights		54	-	2	1	1	1	1	56				
123,086 19,739 10,799																		497 151 9,744 72 83 113 179 764				

1/ Years marked with an asterisk are those in which there were no major accidents, or in which no accidents occurred.

Table 2.- Gas and Dust Explosion Fatalities in Illinois Coal Mines, 1883 to 1932, inclusive.

Years	Fatalities per million		Percentage of explosion fatalities to total fatalities.
	Tons	Man-hours	
1883-1890	0.17	0.04	2.0
1891-1895	.10	.02	2.7
1896-1900	.04	.02	1.3
1901-05	.62	.23	14.5
1906-10	.42	.17	8.7
1911-15	.33	.19	13.0
1916-20	.33	.18	12.4
1921-25	.30	.22	13.2
1926	.26	.13	12.1
1927	.50	.32	21.7
1928	.50	.34	20.1
1929	.16	.12	9.2
1930	.06	.04	2.7
1931	.11	.09	5.5
1932	1.64	1.48	49.1

TABLE 3.- Causes of Explosions

Causes	Number of explosions	Percent
Open lights	27	47.5
Blown-out shots of black blasting powder	12	21.0
Mine fires	6	10.5
Electricity	3	5.3
Explosives	4	7.0
Striking matches	3	5.2
Total	57	100.0

Open lights.- Although open lights were responsible for 47.5 percent of the explosions in Illinois, during the 7-year period 1926 to 1932, inclusive, there were 8 explosions, and only 1 is attributed to open lights. It would therefore appear that general adoption of electric cap lamps in the gassy mines of the southern field has materially reduced the number of explosions from this cause.

Blown-out shots of black blasting powder.- Reckless use of black blasting powder, including pellet powder, caused 12 explosions of more or less violence and extent, in which 48 men were killed. In addition, 115 minor explosions from blown-out shots, caused 179 fatalities; thus 227 miners, mostly shot firers, lost their lives as a result of the use of black blasting powder to "bring down" coal in Illinois mines.

Mine fires.- Six explosions occurred as a result of mine fires, 3 at 1 mine alone. At this particular mine 3 men were killed in the first explosion, 5 in the second (which took place 2 days after the first), and 1 man on top in the third (which occurred on the fourth day after the first). The first 2 explosions occurred during attempts to seal the fire underground.

In one instance, during the sealing of a fire 21 men were killed by the ignition of gas in the return from the fire by an open light some distance outby, where the seals were being built.

Electricity.- Five explosions were due to electricity; the first one occurred in 1917 and the others in 1927, 1929, 1930, and 1931. This emphasizes the increased explosion hazard that has been introduced into the mines of this State by the very marked increase in the use of electrically operated mine equipment in the gassy and dusty mines of southern Illinois in recent years and points clearly to the desirability of increased precautions against gas accumulations through sluggish or interrupted ventilation, as well as to the need for use of permissible mine equipment.

Explosives.- Four explosions were initiated by accidental ignition of black blasting powder. However, none of these took place in recent years, indicating that greater care is now being used in handling and transporting black blasting powder than formerly, and also that the increased use of permissible explosives has reduced this hazard materially.

Striking matches.- Striking matches in closed-light mines caused three explosions; in two instances men attempted to smoke and in the third a brattice man tried to light his flame safety lamp while in an explosive mixture of gas. These explosions indicate the desirability of rules against carrying matches and smoking materials into closed-light mines and strict enforcement of such rules by searching the employees at intervals.

The explosion caused by attempting to relight a flame safety lamp with a match shows the necessity for using only permissible flame safety lamps, which may be unlocked only with a suitable magnet and which are equipped with igniters for lighting, making it unnecessary to take them apart to light them with a match should the flame be accidentally extinguished.

Types of Explosions in Illinois Mines

Table 4 classifies, in accordance with available data, the mines in which explosions occurred as gassy and "nongassy" and shows the number and types of explosions that occurred under each classification.

TABLE 4.- Types of Explosions in Illinois Mines

Condition	Gas	Dust	Gas and dust	Total
"Non-gassy"	11	14	2	27
Gassy	10	2	18	30
Explosions	21	16	30	57
Fatalities	118	67	312	497

CAUSES OF GAS ACCUMULATIONS AND DUST SUSPENSIONS IN ILLINOIS COAL MINES

In 41 of the 57 explosions gas ignitions were the primary cause; in the remaining 16, so far as is known, gas apparently played no appreciable part. Of the 16 explosions in which gas is thought to have played no part 12 were due to blown-out shots of black blasting powder in solid shooting mines. One was due to the ignition by an electric arc from the trolley wire of a cloud of coal dust caused by a wrecked trip; in this mine a dip or "swag" in the haulage entry caused the car couplings in the trip alternately to slacken and tighten, resulting in bumping of cars with considerable coal spillage at this point; the coal was ground up under the car wheels, causing a large amount of dust; it was at this point that the wreck occurred, knocking out several sets of timbers. It is noteworthy, however, that the entry was well rock-dusted on either side of this swag, and the explosion was stopped in both directions by the rock-dust.

PREVENTION OF EXPLOSIONS IN ILLINOIS MINES

In the reports of the various explosions occurring in Illinois recommendations were made, which it was believed, would prevent a recurrence. These recommendations and the number of times they were made are given in table 6.

Inspection.- Inspection for gas 6 or more hours before the working shift entered the mine, with the possibility of gas accumulations between the time of inspection and the time the shift entered the mine, has been responsible for a number of explosions. Inadequate inspection, or failure to inspect old workings before permitting men to enter them with open lights, has also caused several explosions. Recommendations for better inspection for gas, both before and during the shift, were made 21 times.

One explosion in which 4 men were killed was due to an open light; the fire boss, who carried the open light in addition to a flame safety lamp, encountered a larger body of gas than he expected and ignited it with his open light. This practice is not at all uncommon in mines considered only slightly gassy but is always dangerous, as records show that not fewer than 12 fire bosses and 5 mine foremen have been killed in Illinois mines from this cause, in addition to the case cited above.

TABLE 5.- Causes of Gas Accumulations and Dust
Suspensions in Illinois Mines

Causes	Number of explosions
Insufficient or interrupted ventilation.....	22
Insufficient ventilation at face.....	10
Insufficient ventilation, other than at face.....	9
Interrupted ventilation.....	1
Raised curtain.....	1
Repairing stopping.....	1
Open doors.....	3
Door left open.....	3
Fans.....	3
Fans not operating.....	3
Falls of roof or coal.....	5
Falls of roof released gas.....	1
Pillar removal.....	2
Fall of roof pushed gas out of abandoned workings.....	2
Feeders and inrush of gas.....	1
Inrush of gas from idle adjoining mine.....	1
Miscellaneous.....	7
Removing fire seals before fire extinguished.....	2
Sealing mine fires.....	5
Dust raised into suspension.....	16
Shooting off solid.....	12
Trip of cars wrecked.....	1
Ignition of black blasting powder.....	3
Total.....	57

TABLE 6.- Recommendations for preventing explosions at the time of occurrence

Recommendations	Times recommended
Better inspection	21
Better supervision.	20
Undercutting coal	19
Closed lights	16
Dust control.	15
Better ventilation.	14
Permissible explosives.	13
Incombustible stemming.	3
Permissible equipment	7
Taking explosives into mine when electricity is cut off	3
No smoking.	3
Better placement of shots	2
Leave fire sealed until extinguished.	2
Proper sealing of old workings.	2
Sealing mine on top	1
Shubbing of coal.	1
Remove machine cuttings before shooting	1
Inspection idle workings.	1
Fire boss not to carry open light	1
Reversal of air currents.	1
Permissible flame safety lamps.	1
Not move gas when men are in mine	1
Fencing old workings known or suspected of containing gas	1

Supervision.- Recommendations for better supervision in connection with ventilation, handling of explosives, shooting, and rock-dusting appear 20 times and undoubtedly could or should have been made in many additional instances.

Undercutting coal.- Recommendations for undercutting the coal and discontinuing solid shooting appear 19 times. When the number of fatalities (225) from explosions due to solid shooting is considered the importance of this recommendation is apparent.

Dust control (rock-dusting, sprinkling, and water on cutter bar).- The control of the dust hazard by rock-dusting and watering was recommended 15 times. The efficacy of rock-dusting has been recognized for a number of years, but there has evidently been a general lack of knowledge of the amount of rock-dust and the surfaces on which it should be placed necessary to prevent the propagation of a coal-dust explosion. Since most explosions start in the active working sections of a mine it is absolutely necessary that first consideration be given to these sections rather than to rock-dusting the haulage roads only, as is usually done. The rock-dust should be applied to all open workings, including rooms, entries, and air courses, as well as to the haulage roads. Open and abandoned sections of the mine in which it is impossible to use a rock-dust distributor should be protected by suitable rock-dust barriers.

To control the dust hazard effectively, in addition to rock-dusting the formation of dust should be prevented at its source, namely the face, by the application of water sprays to the mining-machine cutter bars, by wetting down broken coal after shooting, and by sprinkling the roof, ribs, and floor approximately 50 feet back from the face before shooting.

Closed lights.- Recommendations for closed lights appear 16 times in the more recent reports, as it is only within recent years that a satisfactory portable electric cap lamp has been available. Most gassy mines of southern Illinois are now using this type of lamp, a practice that unquestionably has aided materially in the avoidance of explosions in those mines.

Ventilation.- Recommendations for better ventilation appear 14 times; ventilation is perhaps the most important phase of operation of coal mines from an explosion-prevention standpoint. Ventilating fans at most mines are adequate; the greatest drawbacks to proper ventilation at the face workings usually are inadequate air courses and leaky doors and stoppings. The air courses become so choked by falls that it is impossible to pass the desired quantity of air through them, while it is not uncommon for as much as 80 per cent of the air being delivered by the fan to leak from the intake to the return long before the face workings are reached.

Permissible explosives.- Recommendations for the use of permissible explosives appear 13 times; there is no doubt that the increased use of this explosive in preference to black blasting powder has been an important factor in reducing the number of explosions in later years.

Incombustible stemming.- It is relatively common practice to use fine coal dust for tamping shots, although coal-dust stemming increases many times the flame given off by a shot. An explosive that would not ignite gas or dust if incombustible stemming were used may do so if coal dust is used. Recommendations for the use of incombustible stemming were made 8 times.

TYPES OF EXPLOSIVES USED

Table 7 shows the amount of black blasting powder and permissible explosives used in Illinois for 1915, 1919, 1924, 1929, and 1932. The very marked decrease in the use of black blasting powder and the increase in permissibles is apparent.

TABLE 7.- Amount of explosives used, 1915-32, pounds

Year	Black blasting powder	Permissible explosives
1915	27,683,150	1,342,334
1919	40,050,200	2,461,331
1924	39,746,475	3,299,727
1929	10,356,625	3,910,677
1932	5,904,372	1,538,286

From information in the annual reports of the Illinois Department of Mines and Minerals.

Permissible equipment.— The use of electrical equipment underground introduces numerous sources of gas and dust ignitions. Bureau of Mines Circular 6540 shows that during the 4 years ended June 30, 1931, 45.8 percent of all mine explosions were of electrical origin and that such explosions accounted for 74.8 percent of all explosion fatalities; during this period in Illinois 3 of the 4 explosions were due to the ignition of gas or dust by electric sparks or arcs.

Recommendations suggesting the use of permissible electrical equipment were made 7 times. In this connection, however, it is recognized that with the depressed condition that has obtained in the coal industry for a number of years it would be financially impossible to replace the present equipment with permissible equipment; however, it would appear to be possible to make it a rule that all new equipment purchased be of the permissible type, and this is being done in a few instances.

Miscellaneous recommendations.— Fourteen additional recommendations were made, but these are covered in a general way by the recommendations that have already been discussed.

FATALITIES AND MINE EXPLOSIONS

Table 8 shows the number of explosions and of fatalities for various periods from 1883 to 1932; it will be seen that the period from 1910 to 1919 marks the high point both in the number of explosions and in the number of fatalities; this is also the period of the most active development of the coal industry in Illinois.

TABLE 8.— Number of explosions and fatalities from
Explosions in Illinois Mines 1883 to 1932

Years	Explosions	Fatalities	Years	Explosions	Fatalities
1883-1889 ^{1/}	1	10	1910-1919	23	176
1890-1899	1	4	1920-1929	14	110
1900-1909	15	138	1930-1932 ^{2/}	3	59

^{1/} 7 years. ^{2/} 3 years

The United States Bureau of Mines, through its Mine Safety Board, issues "decisions" on safety methods and practices. Such decisions as relate to explosion-prevention methods are as follows:

Decision 1:

The United States Bureau of Mines recommends:

1. In all coal mines the portable lamps for illumination be permissible, portable, electric mine lamps; and also
2. In places where fire damp or black damp is liable to be encountered, a permissible magnetically-locked flame safety lamp for gas detection, or equivalent permissible device, be supplied to at least one experienced employee in each such place; and

3. Any employee before being supplied with a permissible flame safety lamp be examined by a competent official of the mine to assure the man's ability to detect gas; and

4. All coal mines whether classed as nongassy or gassy in any part, be supplied with magnetically locked, permissible, flame safety lamps, properly maintained and in sufficient number for all inspection purposes.

Decision 2:

In the interests of safety the United States Bureau of Mines recommends that for blasting in coal mines, permissible explosives, fired electrically, be exclusively used; and that as an aid to blasting, all coal which is feasible to cut, should be cut or sheared.

Decision 4:

In the interest of safety, the United States Bureau of Mines, recommends that auxiliary fans or blowers should not be used in coal mines as a substitute for methods of regular and continuous coursing of the air to every face of the mine.

Decision 5:

To prevent the propagation of mine explosion, the United States Bureau of Mines recommends rock-dusting all coal mines, except anthracite mines, in every part, whether in damp or dry condition. It also recommends that rock-dust barriers be used to sectionalize the mine as additional defense; but these should not be regarded as a substitute for generalized rock-dusting.

Decision 6:

In the interest of safety, the United States Bureau of Mines recommends that in coal mines all entries, rooms, panels, or sections that cannot be kept well ventilated throughout or cannot be inspected regularly and thoroughly, or that are not being used for coursing the air, travel, haulage, or the extraction of coal, be sealed by strong fireproof stoppings.

Decision 7:

In the interest of safety, the United States Bureau of Mines recommends:

1. That the main intake and main return air currents in mines be in separate shafts, slopes, or drifts.
2. That the main intake shaft lining be of fireproof construction, and there be a minimum amount of inflammable material in or adjacent to the shaft.

Decision 9:

The United States Bureau of Mines recommends in coal-mine ventilation practice the following specifications as to unit quantity and quality of air:

1. The quantity in cubic feet of pure intake air flowing per minute in any ventilating split shall be at least equal to 100 times the number of men in that split.

2. The quantity of air entering each unsealed place shall be at least 200 cubic feet per minute and as much more as may be necessary to properly dilute and carry away inflammable or harmful gases which may be present.

3. The air shall be made to circulate continuously to the face in every unsealed place into which an appreciable amount of methane enters.

4. The air in any unsealed place shall be considered unfit for men if it shall be found to contain less than 19 percent oxygen (dry basis), more than 1 percent carbon dioxide or a harmful amount of poisonous gas.

5. If the air in any unsealed place, when sampled or tested in any part of that place not nearer than 4 feet from the face and 10 inches from the roof, shall be found to contain:

(a) more than $1\frac{1}{2}$ percent of inflammable gas, the place shall be considered to be in hazardous condition and require improved ventilation, and

(b) if more than 2 $\frac{1}{2}$ percent of inflammable gas is found, the place shall be considered dangerous, and only men who have been officially designated to improve the ventilation and are properly protected shall remain in or enter said place.

6. If the air in the split which ventilates any group of workings contains more than $1\frac{1}{2}$ percent of inflammable gas, these workings shall be considered to be in a dangerous condition and only men who have been officially designated to improve the ventilation and are properly protected, shall remain in or enter said workings.

Decision 11:

In the interest of safety, the United States Bureau of Mines recommends that in coal mines, haulage and (or) hoisting be kept in intake air as far as possible.

Decision 12:

The United States Bureau of Mines extending Mine Safety Decision No. 2, recommends that for blasting either coal or rock in coal mines, permissible explosives or equivalent permissible device be used exclusively, and in addition, recommends that in blasting--

1. Each charge shall be in a hole properly drilled and stemmed with incombustible material.

2. Each shot shall be fired separately by a permissible single-shot blasting unit, using an electric detonator or igniting equivalent of a kind specified by the bureau for the particular permissible explosive or permissible blasting device.

3. Before and following each shot in gassy and slightly gassy coal mines, examination for gas shall be made with a permissible flame safety lamp or permissible equivalent and--

4. If more than $1\frac{1}{2}$ percent of inflammable gas is found, in the quantity and by the method specified in Mine Safety Decision No. 9, the place shall be considered to be in a hazardous condition and before another shot is fired the gas shall be reduced by ventilation below the percentage and quantity specified in decision 9.

5. Each shot employing explosives shall be prepared and fired by or under the immediate supervision of a man having a state certificate as a mine examiner, fire boss, or foreman; and whenever conditions permit all other men than those authorized to prepare and fire shots shall be out of the mine when shot firing with explosives is being done.

Decision 13:

The United States Bureau of Mines recommends that when electricity is used in coal mines rated as gassy, or wherever in any mine the atmosphere may become gassy:

1. Electrical equipment shall be permissible.
2. Nonpermissible electrical equipment shall be used only in pure intake air.
3. Electrical power shall be cut off whenever the air in the workings is in a dangerous condition, due to inflammable gas.

Decision 15:

In the interest of safety in coal mining, the United States Bureau of Mines to lessen the coal-dust explosion hazard recommends that:

1. Machine coal cuttings be wet as the cutting is being done.
2. The coal face, and the working place 40 feet therefrom, shall be kept free of coal-dust by the use of water.
3. The top of loaded cars in the working place shall be wet.

Decision 16:

In the interest of safety in coal mining, the United States Bureau of Mines recommends that:

1. Machine cuttings be removed from the cut.
2. If the machine cuttings are of a character which would contribute to a dust explosion, they shall be sent out of the mine.

Decision 17:

To lessen the formation and distribution of coal dust in haulage-ways, the United States Bureau of Mines recommends that in bituminous and lignite coal mines:

1. The mine cars should be constructed and maintained dust-tight.
2. The coal should be so loaded that it will not shake off in haulage.
3. The cars and loads should be so sprayed as to prevent dust being distributed along the haulage ways.

Decision 18:

In the interest of safety in coal mining, the United States Bureau of Mines recommends that:

1. The foreman regularly in charge of underground operations and also any person who, in the absence of the foreman, may be placed in temporary charge should each have a certificate of competency from the State to act as mine foreman.

2. The superintendent or person in responsible charge of the mine, to whom the mine foreman reports, should have a certificate of competency from the State which should be issued upon a showing of underground experience for a period of time as long as that required for a foreman's certificate and upon passing an examination including all technical questions asked in the examination required of foremen.

3. These certificates should expire after some stated period of time, such as 5 years, and should be renewed only after the applicant has again passed the examination required by the State.

Decision 20:

In the interest of safety in underground mining, the United States Bureau of Mines recommends that:

While driving tunnels or drifts and sinking or raising shafts or slopes, and also in their operation, there should be an adequate ventilating current wherever men work or travel.

CONCLUSIONS

The principal underlying causes of mine explosions in Illinois have been insufficient or interrupted ventilation and the reckless use of black blasting powder. The chief causes of ignitions of gas or dust were open lights, blown-out shots of black blasting powder, mine fires, electricity, explosives, and striking matches.

Explosion prevention depends largely upon adequate ventilation and inspection, thorough rock-dusting, the moistening of coal dust at the face to prevent its dissemination throughout the mine, the use of permissible explosives and equipment, and last but by no means least, proper supervision to insure that the foregoing fundamental requirements are kept in effect.

INFORMATION CIRCULAR
UNITED STATES BUREAU OF MINES

WYOMING COAL-MINE EXPLOSIONS - 1881 TO 1931^{1/}

By G. M. Kintz^{2/}

The data in this circular were taken from publications of the United States Bureau of Mines and published annual reports of the Wyoming State Inspector of Coal Mines. The tables included are complete only with regard to major disasters; the Bureau of Mines considers a major disaster one in which 5 or more men are killed. However, minor disasters and gas ignitions recorded in these publications are included. It is known that a number of gas ignitions have occurred of which no record is available.

This is one of a series of papers by the Bureau of Mines giving by States available data regarding coal-mine explosions. This information is published in condensed form in the belief that it may be of value to mine operators in preventing future coal-mine explosions.

Explosions and gas ignitions.- Tables 1 and 2 give available data on number killed and injured, the cause and possible means of prevention at the time of 68 explosions and gas ignitions that occurred in Wyoming from 1881 to 1931, inclusive.

TABLE 1. - Wyoming coal-mine explosion data, 1881 - 1931

Year	No. of explosions of gas ignitions	No. killed	No. injured	Year	No. of explosions or gas ignitions	No. killed	No. injured
1881	1	38		1918	4		5
1886	1	13		1919	3	2	2
1895	1	60		1920	7		10
1901	1	22		1921	6		8
1903	1	169		1922	10		16
1905	1	18		1923	12	99	12
1907	1	1	31	1924	2	39	1
1908	1	59		1926	2	2	1
1911	2	1		1927	2	1	4
1912	1	7		1929	1		1
1917	5		7	1931	3		5
Total, 1881 - 1931					68	531	103

^{1/} The Bureau of Mines will welcome reprinting of this paper, provided the following footnote acknowledgment is used: "Reprinted from U. S. Bureau of Mines Information Circular 6765."

^{2/} Associate mining engineer, U. S. Bureau of Mines, Denver, Colo.

TABLE 2. - Explosions and gas ignitions in Wyoming coal mines,
1881 - 1931

Case	Date	County	Type of opening	Production, tons per year	No. employed	No. in mine	Extent of explosion	Cause of presence of explosive atmosphere	Probable source of ignition	Possible means of prevention at time	Gas or dust explosion	No. killed	No. injured	No. escaped uninjured
1	3/4/1881	Uintah	Slope	-	-	-	-	-	-	-	-	38	-	-
2	1/13/1886	do	do	-	-	-	-	-	-	-	-	13	-	-
3	3/20/1895	do	do	-	65	60	-	-	-	-	Gas and dust	60	-	-
4	10/26/1901	Lincoln	do	-	-	-	-	Said to be gas. Short circuit of air while fighting mine fire.	Said to be mine fire.	-	Gas	22	-	-
5	6/30/1903	Carbon	do	-	-	-	-	-	Blown-out shot.	-	Gas and dust	169	-	-
6	12/1/1905	Lincoln	do	-	-	-	-	-	-	-	-	18	-	-
7	1907	Sweetwater.	do	-	-	32	Localized	Runaway trip down slope wrecked raising cloud of dust.	Open lights.	Sprinkling of trip and use of derailing device.	Dust	11/	31	-
8	3/8/1908	Carbon	do	-	-	59	Entire mine	-	Mine fire.	-	Gas and dust	59	-	-
9	1/27/1911	Sheridan	Drift	312,094	285	-	-	Gases distilled by mine fire.	Mine fire due to spontaneous combustion.	Better system of development.	Gas and dust	0	0	0

1/ Died later as a result of injuries received.

TABLE 2. - Explosions and gas ignitions in Wyoming coal mines,
1881 - 1931--Continued

Case	Date	County	Type of opening	Production, tons per year	No. employed	No. in mine	Extent of explosion	Cause of presence of explosive atmosphere	Probable source of ignition	Possible means of prevention at time	Gas or dust explosion	No. killed	No. injured	No. escaped uninjured
10	5/27/1911	Lincoln	Slope	293,355	282	-	-	Unknown.	Gas watchman relighting his lamp.	-	Gas	1	-	-
11	1/20/1912	do	do	146,281	-	-	-	Dust cloud raised by shots.	Blown-out shot	Better location of holes, permissible explosives	Dust	7	-	-
12	3/23/1917	do	do	-	-	-	-	-	Lighted gas. Cause not given.	-	Gas	0	2	-
13	4/10/1917	do	do	-	-	-	-	-	do	-	do	0	1	-
14	4/15/1917	do	do	-	-	-	-	-	Lighted gas in a hole. Cause not given.	-	do	0	1	-
15	5/28/1917	do	do	-	-	-	-	-	Lighted gas. Cause not given.	-	do	0	2	-
16	9/9/1917	do	do	-	-	-	-	-	do	-	do	0	1	-
17	7/22/1918	Uintah	do	51,147	44	31	-	-	Explosion of gas. Cause not given.	-	do	0	2	-

TABLE 2. - Explosions and gas ignitions in Wyoming coal mines,
1881 - 1931---Continued

Case	Date	County	Type of opening	Production, tons per year	No. employed	No. in mine	Extent of explosion	Cause of presence of explosive atmosphere	Probable source of ignition	Possible means of prevention at time	Gas or dust explosion	No. killed	No. injured	No. escaped uninjured
18	7/29/1918	Sheridan	Slope	326,931	2292/	1782/	-	-	Explosion of gas. Cause not given.	-	Gas	0	1	-
19	8/23/1918	Lincoln	do	-	-	-	-	-	Lighted gas. Cause not given.	-	do	0	1	-
20	10/1/1918	do	do	199,340	-	-	-	-	Ignition of gas. Cause not given.	-	do	0	1.	-
21	1/24/1919	do	-	140,270	-	-	-	-	do	-	do	0	1	-
22	6/3/1919	Sheridan	Shaft	185,754	94	7	Localized	Door left open, short-circuiting air.	Unknown.	Positive ventilation.	Gas and dust	2	0	5
23	6/7/1919	Lincoln	Slope	140,270	-	-	-	-	Ignition of gas. Cause not given.	-	Gas	0	1	-
24	3/26/1920	do	do	171,668	-	-	Localized	Pocket of gas.	Cause not given.	-	do	0	1	-
25	4/5/1920	do	do	209,301	-	-	do	-	do	-	do	0	1	-
26	6/29/1920	do	do	209,301	-	-	do	-	do	-	do	0	1	-
27	7/9/1920	Sheridan	Shaft	250,639	213	-	do	Pocket of	do	-	do	0	1	-
28	8/10/1920	Lincoln	Slope	160,451	-	-	do	-	-	-	do	0	1	-
29	12/18/1920	do	do	171,668	-	-	do	-	Coal-dust flame. Cause not given.	-	Dust	0	4	-
30	12/24/1920	do	do	25,149	-	-	-	-	do	-	do	0	1	-

2/ Includes two mines.

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(1608)

TABLE 2. - Explosions and gas ignitions in Wyoming coal mines,
1881 - 1931--Continued

Case.	Date	County	Type of opening	Production, tons per year	No. employed	No. in mine	Extent of explosion	Cause of presence of explosive atmosphere	Probable source of ignition	Possible means of prevention at time	Gas or dust explosion	No. killed	No. injured	No. escaped uninjured
31	2/11/1921	Lincoln	Slope	197,410	-	-	Localized	-	Cause not given.	-	Gas	0	2	-
32	6/18/1921	do	do	197,410	-	-	do	-	do	-	do	0	1	-
33	8/23/1921	do	do	81,423	-	-	do	-	Ignition of gas. Cause not given.	-	do	0	1	-
34	9/26/1921	Lincoln	do	256,107	-	-	do	-	do	-	do	0	1	-
35	10/18/1921	Sweetwater	do	69,960	-	-	do	-	do	-	do	0	2	-
36	11/8/1921	Lincoln	do	197,410	-	-	do	-	do	-	do	0	1	-
37	1/27/1922	do	do	188,931	-	-	do	-	do	-	do	0	1	-
38	1/30/1922	do	do	115,717	-	-	do	-	do	-	do	0	5	-
39	8/14/1922	Sweetwater	do	70,961	-	-	do	-	do	-	do	0	1	-
40	9/6/1922	Lincoln	do	188,931	-	-	do	-	do	-	do	0	1	-
41	9/19/1922	do	do	115,717	-	-	do	-	do	-	do	0	1	-
42	10/12/1922	do	do	115,717	-	-	do	-	do	-	do	0	1	-
43	11/5/1922	Carbon	do	250,087	-	-	do	-	Open light.	Better inspection by fire boss.	do	0	1	-
44	11/28/1922	Sheridan	do	88,688	207	-	do	Pocket of gas in high place. Struck pocket of gas, igniting it.	do	-	do	0	2	-
45	12/20/1922	Lincoln	do	115,717	-	-	do	-	Ignition of gas. Cause not given.	-	do	0	1	-

TABLE 2. - Explosions and gas ignitions in Wyoming coal mines,
1881 - 1931--Continued

Case	Date	County	Type of opening	Production, tons per year	No. employed	No. in mine	Extent of explosion	Cause of presence of explosive atmosphere	Probable source of ignition	Possible means of prevention at time	Gas or dust explosion	No. killed	No. injured	No. escaped uninjured
46	12/21/1922	Lincoln	Slope	188,931	-	-	Localized	Pocket of gas.	Ignition of gas. Cause not given.	-	Gas	0	2	-
47	1/18/1923	do	do	163,549	-	-	-	-	do	-	do	0	1	-
48	1/30/1923	do	do	163,549	-	-	-	-	do	-	do	0	1	-
49	2/18/1923	do	do	163,549	-	-	-	-	do	-	do	0	1	-
50	2/24/1923	do	do	178,863	-	-	-	-	do	-	do	0	1	-
51	5/12/1923	do	do	163,549	-	-	-	-	do	-	do	0	1	-
52	7/7/1923	do	do	163,549	-	-	-	-	do	-	do	0	1	-
53	7/19/1923	do	do	178,863	-	-	-	-	do	-	do	0	1	-
54	7/21/1923	do	do	163,549	-	-	-	-	do	-	do	0	1	-
55	8/14/1923	do	do	201,253	-	135	Entire mine, not violent	Uncertain; probably line brattice de-ranged by a fall.	Fire boss attempting to relight flame safety lamp in presence of gas.	Use of magnetically-locked flame safety lamp, rock-dust; better ventilation.	Gas and dust	99	2	36
56	9/25/1923	Sweet-water	do	98,584 ^{3/4}	-	-	-	-	Ignition of gas. Cause not given.	-	Gas	0	1	-
57	9/27/1923	Lincoln	do	163,549	-	-	-	-	do	-	do	0	1	-
58	10/6/1923	do	do	178,863	-	-	-	-	do	-	do	0	1	-
59	3/10/1924	Campbell	Shaft	19,263	50	-	Localized	-	-	-	do	0	1	-

^{3/4} Includes production of two mines.

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(1608)

TABLE 2. - Explosions and gas ignitions in Wyoming coal mines,

1881 - 1931--Continued

Case	Date	County	Type of opening	Production, tons per year	No. employed	No. in mine	Extent of explosion	Cause of presence of explosive atmosphere	Probable source of ignition	Possible means of prevention at time	Gas or dust explosion	No. killed	No. injured	No. escaped uninjured
60	9/15/1924	Lincoln	Slope	124,994	-	-	Entire mine	Leaky stoppings and brattices.	Electric arc.	Increased ventilation; tight stoppings; better installation of line of brattice.	Gas	39	0	12
61	6/9/1926	Fremont	do	76,456	89	-	Localized	-	Ignition of gas. Cause not given.	-	Gas	0	1	-
62	6/26/1926	Uintah	do	-	-	2	Entire mine	Shot of black blasting powder raised dust cloud.	Blown-out shot.	-	Dust	24/	0	-
63	6/9/1927	Lincoln	do	83,508	92	20	Localized	Open "slant" door short-circuited the air.	Arc from trolley.	Permissible storage-battery locomotives.	Gas	1	3	-
64	10/22/1927	Hot Springs	do	87,382	125	-	-	-	Unknown.	-	do	0	1	-
65	4/23/1929	Sweetwater	do	107,622	97	2	Localized	Feeder opened by shot.	Open light.	Permissible lamps.	do	0	1	-

4/ Includes 1 visitor.

TABLE 2. - Explosions and gas ignitions in Wyoming coal mines,
1881 - 1931--Concluded

Case	Date	County	Type of opening	Production, tons per year	No. employed	No. in mine	Extent of explosion	Cause of presence of explosive atmosphere	Probable source of ignition	Possible means of prevention at time	Gas or dust explosion	No. killed	No. injured	No. escaped uninjured
66	5/26/1931	Lincoln	Slope	90,719	85	2	Localized	Pocket near roof.	Electric arc from non-permissible electric drill.	Better ventilation; permissible electric drill.	Gas	0	2	-
67	6/24/1931	do	do	90,719	85	7	do	Brattice down.	Electric arc from non-permissible mining machine.	Better care of ventilation.	do	0	2	-
68	10/12/1931	Uintah	do	12,462	22	20	do	Came from crevice. Old workings being reopened.	Open light.	Closed lights.	do	0	1	-

Nature of explosions.- Approximately 72 percent of the men killed in the explosions listed in this paper lost their lives in the 8 explosions that occurred before 1910. The nature of the explosions with reference to gas and coal dust is shown in table 3:

TABLE 3. - Nature of explosions, with reference to gas or coal dust

Nature of explosion	No. of explosions	No. killed	No. injured
Unknown	3	69	0
Gas	53	24	67
Dust	5	10	36
Gas and dust	7	428	0
Total	68	531	103

In the 53 listed gas explosions 24 persons were killed and 67 injured. In the 5 listed dust explosions 10 men were killed and 36 injured. In the 7 listed gas and dust explosions 428 men were killed.

These figures illustrate that where a combination gas and dust explosion occurs the destruction of life and property is likely to be much greater than where gas alone is the exploding medium. They show the necessity of thorough rock-dusting and sprinkling of the working places, and of trips where possible, to prevent the participation of coal dust in explosions in bituminous and sub-bituminous mines. A study of widespread coal-mine explosions in the United States shows that almost invariably they involved either dust or both gas and dust.

TABLE 4. - Sources of ignition

Cause	No. of explosions	No. killed	No. injured
Unknown (no record)	51	131	60
Open lights	5	1	36
Relighting flame safety lamp	2	100	-
Mine fires	2	81	-
Spontaneous combustion	1	0	-
Electric arc	4	40	7
Blown-out shot	3	178	-
Total	68	531	103

Open lights.- Probably most of the explosions listed as unknown were caused by the use of open-flame lights, although the records are not clear on this point.

Many persons have expressed the opinion that it is safe to use open lights in so-called nongassy mines and that there is only slight chance of ignition of a dust cloud thereby. Case 7 illustrates the fallacy of this opinion. Here the last trip of the shift broke loose from the tippie, ran down the slope, and was wrecked near a point where 32 miners wearing open lights were waiting for the man trip. The dust raised by this wrecked trip was ignited by the miners' open lights, killing 1 and injuring many of the others. Had the mine not been relatively clean at this place, the violence of the explosion probably would have been sufficient to kill all the men. The United States Bureau of Mines recommends (see Mine Safety Board Decision 1 and discussion thereof in this paper) portable closed lights of permissible type in all coal mines, not only from the hazard of dust ignition, as illustrated by this example, but also because of the gas-ignition and fire hazard of the open flame lamp.

In view of past experience with the ignition of explosive gas by open flame lamps and the demonstrated practicability, relatively good illumination, and cheapness of the electric lamp it is hard to understand why many coal-mine operators still permit the use of open lights in their mines.

Flame safety lamps.- In all, 100 men were killed in two explosions because flame safety lamps were taken apart and relighted in the presence of an explosive atmosphere. In both cases the men who committed this act probably knew better, but the human element in accident-prevention work frequently cannot be controlled; therefore, mine equipment should be as nearly "fool proof" as possible. The present up-to-date permissible flame safety lamp is magnetically locked and cannot be opened except by the use of a rather powerful, large magnet. The United States Bureau of Mines has no record that a properly assembled and properly maintained flame safety lamp ever caused an explosion, although numerous gas ignitions have been caused in coal mines both in the United States and in other countries when flame safety lamps have been mishandled; hence the flame safety lamp is by no means "fool proof". With these facts in mind, it is clear why the Bureau of Mines recommends the permissible magnetically-locked flame safety lamp; if this lock is kept in proper working condition and if the magnet is available only outside of the mine there will be little likelihood that a man will open his light underground.

Spontaneous combustion.- Some of the coal mines in Wyoming are subject to spontaneous combustion. The explosion listed as case 9 was caused by a fire started by spontaneous combustion. The fact that no one was killed or injured in this explosion was due to the good judgment of the mine foreman, who withdrew every man from the mine just a few minutes before the explosion occurred. The prevention as well as the successful control of mine fires, especially those due to spontaneous combustion, depends in great part upon the proper development and operation, and especially the cleanliness of the mine.

Mine fires.- During the fighting of mine fires in "soft" coal mines there is always a great gas-ignition hazard; those fighting such fires

should fully realize this and take all possible precautions to protect themselves. If the area of the mine adjacent to the fire is given an extra heavy coating of rock-dust and the rest of the mine is properly rock-dusted, an explosion caused by the accumulation of gases by the mine fire probably will be localized enough to prevent much, if any, loss of life or damage to property.

Electric arc.- The coal mines of Wyoming probably have installed more electrical mining devices proportionately than in any other State in the Union. The increased installation of electrically operated mining machinery has greatly increased the explosion hazard from the ignition by an electric arc of an explosive mixture of gas, dust, or other combustible matter. Many types of permissible coal-mining machinery are available and should be used; it must be remembered, however, that a machine is permissible only as long as it is maintained and operated according to the instructions of the manufacturer and of the United States Bureau of Mines. In general, these instructions will be found on the permissible plate on every permissible machine or piece of equipment. Permissible electrical equipment maintained in permissible condition will greatly reduce the hazard of gas ignitions in places in mines that may become gassy. The Mine Safety Board of the United States Bureau of Mines makes the following recommendation regarding electricity in coal mines in decision 13³/:

The United States Bureau of Mines recommends that when electricity is used in coal mines rated as gassy, or whenever in any mine the atmosphere may become gassy:

1. Electrical equipment shall be permissible,
2. Nonpermissible electrical equipment shall be used only in pure intake air,
3. Electrical power shall be cut off whenever the air in the workings is in a dangerous condition, due to inflammable gas.

Operators and mine electricians will find valuable information regarding electricity in mines in United States Bureau of Mines Technical Paper 402, Safety Rules for Installing and Using Electrical Equipment in Coal Mines.

Blown-out shots.- The use of black blasting powder and dynamite or the improper use of permissible explosives caused at least 3 explosions which resulted in the death of 178 men, approximately one third of all the men killed in the explosions listed in this paper. The poor placement of drill holes, too heavy charges of explosives, insufficient or improper tamping of holes, and blasting off the solid were some of the reasons for blown-out shots.

3/ United States Bureau of Mines, Mine Safety Board Decision 13; Recommendations of the U. S. Bureau of Mines on Certain Questions of Safety: Inf. Circ. 6732, July 1933, p. 22.

TABLE 5. - Causes of gas accumulations or coal-dust clouds

Cause	No. of explosions	No. killed	No. injured
Short-circuiting of air, including leaky stoppings and deranged brattices down	6	163	5
Pocket of gas	6		8
Gases distilled by mine fires	1	0	
Feeder opened by shot	1	0	1
Gas from old workings	1	0	1
Dust raised by shots	2	9	
Dust cloud raised by wrecked trip	1	1	31
Unknown	50	358	57
Total	68	531	103

The explosives now designated as "permissible" were used first in 1907; since that time explosives manufacturers have from time to time improved them until now it is possible to obtain permissible explosives that will give virtually the same blasting results as black blasting powder. Explosives engineers have proved that the production of lump coal does not depend wholly upon the speed of detonation of an explosive; strength and density as well as other factors are also important. A good method of determining the proper explosive to use in blasting any coal is to consult the explosives manufacturers, who have explosives engineers well qualified to recommend the proper grade of permissible explosive.

If permissible explosives are not used according to the methods prescribed by the Bureau of Mines and the explosives manufacturers they may ignite an explosive mixture of gas or a coal-dust cloud; even though the explosive is properly manufactured and approved by the Bureau of Mines, it is not permissible if used in any of the following ways:

1. If stored under improper conditions until it undergoes a change in character.
2. If used in a frozen or partly frozen condition.
3. If used in excess of $1\frac{1}{2}$ pounds per shot.
4. If the diameter of the cartridge is less than that designated in the column "smallest permissible diameter".
5. If fired with an electric detonator of less efficiency than that prescribed.
6. If fired with fuse.

7. If fired without stemming.
8. If fired with combustible stemming.
9. If fired in the presence of a dangerous percentage of fire damp.
10. If the shot is a depending shot, is bored into the solid, or has a burden so heavy that the shot obviously is liable to blow out.

Many mines in Wyoming still use black blasting powder for blasting coal; also many mines in Wyoming do not rock-dust. Dust from the coals in Wyoming, tested by the United States Bureau of Mines, has been readily detonated in the Experimental mine by a blown-out shot of black blasting powder; therefore, unless the element of good luck continues to intervene, the stage is set for a dust explosion in such mines any time the necessary conditions arise.

CAUSES OF ACCUMULATIONS OF EXPLOSIVE ATMOSPHERES

Short-circuiting of air.- It is almost impossible to operate a mine without using some doors, but there is no reason why they should not be in pairs or supplemented with curtains. Doors should be self-closing, should be fireproofed, and should be constructed as nearly airtight as possible; they should not be equipped with latches to hold them open.

In case 4 the air presumably became short-circuited while men were fighting a mine fire. During the handling of a mine fire it is natural for all efforts to be concentrated upon extinguishing the fire; however, the person in charge should pay particular attention to the ventilation so that accumulations of gas will not occur where they can be moved around by the flame from the fire and ignited by it.

When old workings that give off quantities of methane gas are sealed the seals should be inspected regularly to see that they do not leak. A ventilating current should also be made to sweep the face of the seals in such a way as to prevent the accumulation of gas.

Two of the explosions listed under this heading were due to line brattices either being down or improperly installed so that enough air was not reaching the working face to keep it clear. Many mine officials and underground workers do not take the installation of line brattices seriously, especially in so-called nongassy mines. When properly installed with plenty of air available, line brattices are the best means of ventilating working places when advancing between crosscuts or breakthroughs. As there is leakage of air, even with the best-installed curtains and brattices, particular care should be taken by mine officials to see that they are installed to keep the leakage at a minimum.

Pocket of gas.- The inflammable gas commonly encountered in coal mines is known as methane, although there are mines in Wyoming that liberate gases

nearly equivalent in composition to some of those given off by oil wells. These gases are of lighter specific gravity than air and unless diffused in the air currents will accumulate in high points in the roof; fire bosses or other persons examining the mine should take particular care to test high spots for accumulations of such gases. If gas accumulates at such points it can be removed by a brattice cloth installed at the roof or possibly by a pipe running from the high pocket directly into the return. When working places are driven to the rise in gassy mines officials should insist that the men keep the line brattice properly installed close to the face; also that crosscuts are driven at proper intervals and that there is enough air at the working place to keep it clear of gas.

Gases distilled by mine fires.- The explosibility of gases distilled by mine fires is illustrated by the explosions that often occur in coal stoves. In mines explosions due to mine fires often are very violent, with a large amount of inflammable gases and coal dust present. The mining system should be so designed that most possible fires can be readily sealed without danger to operation of the rest of the mine.

In addition to the hazard of an explosion from gases distilled by mine fires it must be remembered that such gases also contain rather high percentages of carbon monoxide, of which less than 0.5 of 1 percent can kill a man in a very few minutes.

Feeder opened by a shot.- In many coal mines it is impossible to tell when a shot may open a gas feeder; for this reason mines should be inspected after places have been shot and before the miners enter. The Wyoming law permits blasting of the coal at any time; the man that fires the shot, therefore, should carry a flame safety lamp and be competent to use it.

Gas from old workings.- In case 68, during the reopening of an old mine explosive gas from old workings leaked through a crevice in the coal; when old mines are reopened great care should be taken to examine the air, especially when old workings are approached, not only because of possibly encountering explosive gas but also from the hazard of black damp. Where the exact location of old workings is not known drill holes should be drilled ahead of the advancing workings and also to the sides; such drill holes should be deep enough so that when the coal is blasted there will be no danger that the shot will break into the old workings.

Dust raised by shots.- Improper location of drill holes, shooting off the solid, firing of dependent shots, use of too little or no stemming, and loading holes too heavily cause blown-out shots; these can be eliminated by proper supervision, assisted by safety education of miners and shot firers. If the shots are loaded and fired when the working shift is out of the mine the likelihood of wholesale fatalities from explosions caused by blown-out shots or from other causes in connection with the use or abuse of explosives in coal mines will be eliminated. In several mines in the United States all shots are fired from the outside of the mine while all the men are out of the

mine. The explosives are taken into the mine by the shot firers after the shift is out of the mine; the shot firer loads and tamps all holes. If this practice is not feasible the next best method is to have a short firer load, tamp, and fire all the shot holes while all other persons are out of the mine.

As the law in Wyoming sanctions blasting the coal at any time during the shift, even in gassy mines, there is little likelihood that these methods will be used to any great extent in Wyoming mines. A good shot firer will not fire a charge placed in an improperly drilled hole, will not blast a place that is not properly undercut or sheared or where the "bug" dust is not removed from the cut, and will not overload a hole or tamp it with stemming containing coal dust. Therefore, the use of a shot firer to handle all explosives and to load and tamp all holes is probably the safest method possible with present practice in Wyoming. The United States Bureau of Mines, in Mine Safety Board Decision 24/, recommends that for blasting in coal mines permissible explosives fired electrically be used exclusively and that as an aid to blasting all coal which is feasible to cut should be cut or sheared. The United States Bureau of Mines further, in its Mine Safety Board Decision 124/, makes the following recommendation:

The United States Bureau of Mines, extending Mine Safety Decision 2, recommends that for blasting either coal or rock in coal mines, permissible explosives or equivalent permissible device be used exclusively, and in addition recommends that in blasting -

1. Each charge shall be in a hole properly drilled and stemmed with incombustible material.

2. Each shot shall be fired separately by a permissible single-shot blasting unit, using an electric detonator or igniting equivalent of a kind specified by the Bureau for the particular permissible explosive or permissible blasting device.

3. Before and following each shot in gassy and slightly gassy coal mines, examination for gas shall be made with a permissible flame safety lamp or permissible equivalent and -

4. If more than $1\frac{1}{2}$ percent of inflammable gas is found, in the quantity and by the method specified in Mine Safety Decision 9, the place shall be considered to be in a hazardous condition and before another shot is fired the gas shall be reduced by ventilation below the percentage and quantity specified in decision 9.

5. Each shot employing explosives shall be prepared and fired by or under the immediate supervision of a man having a state certificate as a mine examiner, fire boss, or foreman; and whenever conditions permit all other men than those authorized to prepare and fire shots shall be out of the mine when shot firing with explosives is being done.

4/ See footnote 3.

Rock-dusting of all places to within a short distance of the working face and thorough wetting of the places from the end of the rock-dusted zone to and including the face before blasting will lessen the likelihood of occurrence of a coal-dust cloud.

Dust cloud raised by wrecked trip.- Coal-dust clouds raised by wrecked trips have caused the death of a number of men in the mines of Colorado and New Mexico. These explosions direct attention to the necessity for properly maintained and inspected haulage equipment and haulageways to prevent the occurrence of disasters of this type. As previously stated, the explosion listed as case 7 could have been prevented if a proper derail had been placed at the head of the slope. Sprinkling of trips, thorough rock-dusting of haulageways, rock-dust barriers, and keeping haulageways clean will tend to prevent the raising of dust clouds by a wreck or will localize the resulting explosion if a dust cloud is ignited.

Unknown.- Many if not most gas ignitions in coal mines are due to defective ventilation caused by improper installation of stoppings, doors, curtains, and line brattices and the use of doors in place of overcasts. The particular fault with the ventilation is not known for 52 of the mines in which the cause of the gas accumulations was not given in the reports available.

MEANS OF PREVENTING EXPLOSIONS AT THE TIME

Table 6 lists the means that would have prevented the explosions had they been in effect at the time.

TABLE 6. - Means of preventing explosion at time

Means of prevention	No. of explosions	No. killed	No. injured
Permissible flame safety lamp	1	99	
Permissible portable electric lights	2		2
Tight stoppings and better installation of line brattice	1	39	
Permissible mining machinery.	2	1	5
Sprinkling trip, derailling device	1	1	31
Better system of development.	1	0	
Better location of drill holes; permissible explosives	1	7	
Better inspection by fire boss	1	0	1
Positive ventilation	1	2	
Better care of ventilation ..	1	0	2
Unknown	56	382	62
Total	68	531	103

Permissible flame safety lamps and permissible portable electric lamps.-
The United States Bureau of Mines^{5/} makes the following decision relating to miners' lamps in coal mines:

1. In all coal mines the portable lamps for illumination be permissible, portable, electric mine lamps; and also
2. In places where fire damp or black damp is liable to be encountered, a permissible, magnetically locked, flame safety lamp for gas detection, or equivalent permissible device, be supplied to at least one experienced employee in such place; and
3. Any employee before being supplied with a permissible flame safety lamp be examined by a competent official of the mine to assure the man's ability to detect gas; and
4. All coal mines whether classed as nongassy or gassy in any part, be supplied with magnetically locked, permissible, flame safety lamps, properly maintained and in sufficient number for all inspection purposes.

The recommendations were made for the following reasons:

Open lamps or so-called safety lamps of unapproved kind may cause fires or ignite firedamp and coal-dust, and therefore they should not be used in coal mines. Miners' electric lamps have been so improved in illumination, convenience, and safety, that it is no longer to be considered a hardship in mining to use a portable permissible electric lamp, in fact it is quite the reverse.

However, since no approved electric lamp shows the presence of methane or indicates oxygen deficiency, one experienced employee in every place where such conditions are liable to be found should be provided with a permissible flame safety lamp, or equivalent permissible device to be used for testing.

Although a permissible flame safety lamp is safe if properly assembled and used, there is a possibility of its being a source of ignition of firedamp if the lamp is illegally opened in the mine or is dropped and the glass broken, or if the lamp is struck by a sharp point or is moved so rapidly as to cause flame to extend through the gauzes. Accordingly the use of the permissible flame safety lamp should be limited to gas testing in the hands of competent and experienced employees who can demonstrate with accuracy and safety the percentage of methane in the air in excess of $1\frac{1}{2}$ or 2 percent. The flame safety lamp also gives information as to the sufficiency of oxygen in the surrounding atmosphere.

^{5/} Mine Safety Board Decision 1. See footnote 3.

The first part of the report deals with the general situation of the country and the progress of the work during the year.

The second part contains a detailed account of the work done in the various departments.

The third part gives a summary of the results of the work and a statement of the financial position.

The fourth part contains a list of the names of the persons who have been employed during the year.

The fifth part contains a list of the names of the persons who have been employed during the year.

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The eleventh part contains a list of the names of the persons who have been employed during the year.

The twelfth part contains a list of the names of the persons who have been employed during the year.

Other underground employees working or traveling in well-ventilated places should be provided with portable permissible electric lamps giving an excellent steady light; the good illumination which these lamps provide if maintained in proper condition tends to prevent accidents from falls and other hazards, and the equipment itself is free from the danger of igniting gas and coal dust.

Tight stoppings, better installation of line brattice.- Leaky stoppings between the air courses and other openings in many coal mines account for the greatest percentage of loss of ventilating current. The money lost from this leakage through excessive consumption of power at the fan may often more than pay for the inspection and repair of these stoppings. The proper installation of line brattice is necessary if mine officials and miners expect the proper ventilation at the working faces. The United States Bureau of Mines makes the following recommendation relating to the sealing of parts of mines which cannot be kept well ventilated and inspected.^{6/}

In the interest of safety, the United States Bureau of Mines recommends that in coal mines all entries, rooms, panels, or sections that cannot be kept well ventilated throughout or cannot be inspected regularly and thoroughly or that are not being used for coursing the air, travel, haulage, or the extraction of coal, be sealed by strong fireproof stoppings.

In the past the question of sealing has been a much disputed one among mining men. Some have considered that it was highly dangerous to inclose areas in a mine which might fill with a body of gas. The Bureau agrees with this point of view, unless the stoppings are strong and fireproof; and by strength is meant sufficient to maintain the stoppings in place if there should be a heavy fall of roof in an empty or partly empty goaf, suddenly compressing the gas or a gas-air mixture.

On the other hand, engineers of the Bureau find from extensive experience that in large room-and-pillar mines in which the pillars are either not extracted at all or not extracted for a long period, the old areas are not generally well-ventilated and often cannot be inspected because of danger of falls, and that the lack of ventilation is inevitable from the enormous territory which may be opened. Furthermore, these regions act as places for accumulating dangerous dust as well as gas.

The Bureau believes that the ideal system of room-and-pillar coal mining is to take out only a small percentage of coal on the advance, generally less than 20 percent, and then to extract the pillars promptly on a systematic line of retreat for the whole mine or for a large panel, which may be considered equivalent to longwall retreating. Under such conditions there should be no need of sealing extensive areas of open unused workings when they can be properly ventilated.

^{6/} Mine Safety Board Decision 6. See footnote 3.

Where the workings of a mine, whether it is a metal or coal mine, have been connected to an adjacent mine and after more or less extraction of the mineral in that part of the mine, the connecting workings are not tightly closed, strong fireproof doors or stoppings should be erected in such connections at or near the boundary of the mine. The objective of this is to prevent fire or fumes entering from one mine to the other. In the case of coal-mine workings which have been connected, these stoppings should not only be fireproof but should be strong enough to resist the pressure of a gas or dust explosion. Explosion tests by the Bureau of Mines on the strength of stoppings indicate that a plain concrete slab acting as a flat arch when properly keyed into the ribs and floor, probably is the most practicable form of construction for a fireproof stopping strong enough to resist an explosion of ordinary degree of violence.

In erection of barrier-pillar stoppings which subsequently may be subjected to high hydraulic pressure through abandonment of the adjacent mine workings and their filling with water, a release to avoid high-water pressure should be provided. This might be an inclined pipe extending down into a water-filled sump on the inby side of the stopping, thus furnishing a water seal to prevent escape of gas into workings in active operation.

Permissible mining machinery.- Permissible coal-mining machinery is available for nearly every operation in or about a coal mine, and the Bureau of Mines recommends^{7/} that in gassy mines electrical equipment shall be permissible.

Sprinkling trip and derailling device.- In many coal mines all loaded trips are sprinkled at the sectional parting and at the main parting. In some mines trips of empty cars are sprinkled before they are allowed to enter the mine. Wetting the coal on a trip or wetting the empty cars assists in preventing the raising of a dust cloud if the trip should be wrecked and prevents the dust from blowing off the trip and settling on ribs, roof, timbers, and road, thus reducing the dust hazard and the necessity of frequent redusting along haulageways.

Better system of development.- Many of the mine explosions and mine fires can be placed under this heading. In general, coal mines should be so developed that if a fire occurs at any point there will be a barrier pillar between it and other workings with a minimum amount of openings so the fire can be completely sealed without excessive danger to the workmen.

Permissible explosives and better location of holes.- As previously stated, the Bureau of Mines recommends the use of permissible explosives only or of equivalent permissible blasting devices. The proper location of drill holes depends much upon the local conditions with respect to the coal bed and it is often difficult to educate the miner to place the holes properly.

^{7/} Mine Safety Board Decision 13. See footnote 3.

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Better ventilation.- Proper ventilation should be given first consideration in the operation or development of any coal mine; rock-dusting and other safety devices are secondary to ventilation.

EXPLOSIBILITY OF WYOMING COAL DUST

The United States Bureau of Mines tests show that semianthracite, bituminous, and subbituminous coal dust is explosive under certain conditions. Experiments by the Bureau of Mines have proved that the ratio of the percentage of volatile matter to volatile matter plus the fixed carbon in the coal, other factors being equal, is an index of the explosibility of coal dust, the higher the volatile matter generally the more explosive the dust. Any coal dust which has this ratio greater than 0.15 is explosive in the absence of gas. The presence of any considerable quantity of inflammable gas in the mine air increases this explosibility of the dust; therefore, an additional quantity of rock-dust should be added to the ribs, roof, floor, timbers, and "caves" in coal mines to neutralize the effect of the gas present. The dust of all Wyoming coals tested at the experimental mine of the United States Bureau of Mines has been found to be explosive; undoubtedly no mine in Wyoming can be considered safe from a dust explosion unless all underground surfaces are thoroughly wet or rock-dusted from the working faces to the mine opening.

CONCLUSIONS

The record of explosions in Wyoming compared to that of neighboring States is relatively good, but Wyoming has had some disastrous explosions and a number of gas ignitions which, although resulting in injury to only one or two men, might have constituted the beginning of disastrous explosions. Many mines still do not use rock-dust or permissible explosives; measures should be taken in all Wyoming coal mines to minimize the possibility of a general explosion by maintaining good ventilation and effective rock-dusting and by using permissible explosives.

INFORMATION CIRCULAR

UNITED STATES BUREAU OF MINES

EXPLOSIONS IN VIRGINIA COAL MINES
1839 TO 1933¹

By Joseph F. Davies² and H. B. Humphrey³

PURPOSE OF THIS REPORT

Analysis of past experience is one of the best guides to improvement, particularly in regard to the prevention of accidents; by this means a relatively true picture of the causes of accidents may be drawn, the need of preventive measures judged, and their effectiveness determined. Explosions are entirely preventable in mining, yet they ranked third among the causes of fatal accidents in the United States during the 10 years ended December 31, 1932, and caused 16.1 percent of the fatal accidents in bituminous-coal mines during that period.

Virginia held eighth place among the States in coal production in 1932. Her mines had an enviable record, with no explosions for several years before 1932; during the first 9 months of that year, however, they had 3 explosion disasters in a period when none occurred in the rest of the coal mines of the United States. This circular gives available information on the explosions in the coal mines of Virginia as far back as 1839. Causes are summarized, the pertinent factors leading to the disasters are detailed and grouped, and the success or lack of protective measures noted.

Data collected and published by the United States Bureau of Mines were used in compiling the tables used in this report; in addition, the Virginia Department of Labor and Industry, through its division of mines, assisted in locating published records. The operating companies in the State freely gave information on explosion occurrences at their properties. Because of the long period of mining and the fact that only summarized tables of fatalities are given in the annual reports, even today, some cases may have been undiscovered; their omission, however, will not materially affect the resulting conclusions.

¹ The Bureau of Mines will welcome reprinting of this paper, provided the following footnote acknowledgment is used: "Reprinted from U. S. Bureau of Mines Information Circular 6766".

² District engineer, U. S. Bureau of Mines, Norton, Va.

³ Assistant engineer, U. S. Bureau of Mines, Norton, Va.

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DEPARTMENT OF CHEMISTRY

RESEARCH REPORT

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1955

REPORT OF

DR. J. H. GOLDSTEIN
AND
DR. R. M. MAYER

ON THE
KINETICS OF
THE
REACTION OF
HYDROGEN
PEROXIDE
WITH
HYDROGEN
SULFIDE

IN
AQUEOUS
SOLUTION
AT
VARIABLE
TEMPERATURES
AND
INITIAL
CONCENTRATIONS

CHICAGO, ILLINOIS

1955

HISTORICAL SUMMARY

The first coal mined in the United States is said to have been along the James River in the Richmond Basin near Richmond, Va. The date usually given is 1750, although some accounts state that this coal was used as early as 1700. The first record of production is for 1822, when 54,000 tons were mined in one year, but after the "War of the 60's" mining declined in this field; although some revival was attempted after 1900, coal production was discontinued in a few years, although great coal reserves remain.

The Pocahontas field was opened in 1822, when the Norfolk & Western Railway was built, and has been an increasingly large producer. Semianthracites^{3a} were mined on a small scale locally in Pulaski and Montgomery Counties from before the Civil War until about 1895; since that time coal has been shipped constantly. The coal fields in Wise and adjoining counties in the Southwest district were opened in 1891 with the building of the Norfolk & Western and the Louisville & Nashville Railroads to Norton.

Thus, by 1894 over a million tons per year were being mined in the State and 2 million by 1899. Production doubled every few years and reached 10 million tons in 1917. The highest figure was 14 million tons, produced in 1926. The output for 1932 was 7,692,000 tons.

The periods of numerous explosions in any of these fields came when the mines were so extended that gas and dust accumulated. In the Richmond Basin this period began when depths were reached that required real ventilation and lasted until work was stopped and the mines filled with water. Likewise, in the semianthracite field of Pulaski and Montgomery Counties explosive gas did not prove troublesome in early years before the mines reached considerable depth and extent.

Drift mines in the Pocahontas and Southwest Virginia districts have had not only local ignitions but also violent explosions in both large and small mines since they were opened.

FATALITIES

A remarkable feature of Virginia mines is the rarity of small explosions resulting in single fatalities; 1 explosion killed 112 men; 2 others killed 55 and 69; 8 had 10 to 40 fatalities each; and the other 12, 2 to 9 fatalities each. This is an abnormally large number of fatalities for the number of explosions involved compared with the experience of other States and may be due to the fact that no record is available of less disastrous explosions which probably occurred in the earlier years, many of which may have resulted in single fatalities. The following tabulation of explosions and deaths for periods of years may prove illuminating:

^{3a} Otherwise known as Virginia hard coal or anthracite mined in Virginia.

The first part of the report deals with the general situation of the country. It is a very interesting and informative study of the country's development. The author has done a great deal of research and has gathered a wealth of material. The report is well written and is a valuable contribution to the study of the country's development.

The second part of the report deals with the economic situation of the country. It is a very interesting and informative study of the country's economic development. The author has done a great deal of research and has gathered a wealth of material. The report is well written and is a valuable contribution to the study of the country's economic development.

The third part of the report deals with the social situation of the country. It is a very interesting and informative study of the country's social development. The author has done a great deal of research and has gathered a wealth of material. The report is well written and is a valuable contribution to the study of the country's social development.

The fourth part of the report deals with the political situation of the country. It is a very interesting and informative study of the country's political development. The author has done a great deal of research and has gathered a wealth of material. The report is well written and is a valuable contribution to the study of the country's political development.

The fifth part of the report deals with the cultural situation of the country. It is a very interesting and informative study of the country's cultural development. The author has done a great deal of research and has gathered a wealth of material. The report is well written and is a valuable contribution to the study of the country's cultural development.

The sixth part of the report deals with the environmental situation of the country. It is a very interesting and informative study of the country's environmental development. The author has done a great deal of research and has gathered a wealth of material. The report is well written and is a valuable contribution to the study of the country's environmental development.

TABLE 1. - Explosions and deaths, by periods

Period	Explosions	No. killed	Average no. killed per explosion
1839 to 1870, 31 years	7	217	31
1871 to 1900, 29 years	5	162	27
1901 to 1930, 29 years	14	98	7
1931	0	0	0
1932	3	54	18
Totals and averages	29	531	18

Table 2 lists the explosions causing fatalities or indicating violence with no accompanying fatalities. Table 3 lists many nonfatal occurrences that might be called local explosions or ignitions. The list is representative of this type of explosions; although not complete, as many such explosives are never recorded, it includes enough of them to show the many potential disasters that were escaped, in most cases by fortunate chance only. In these 42 local ignitions 71 men were more or less seriously burned or injured. Of the 29 explosions given in table 2, resulting in 531 fatalities, 2 caused no deaths and 2 others only 1 death each.

Case	Date	Mine	County	Type of mine	Daily tonnage	Men employed	Explosion			Cause of accumulation	Source of ignition	Other factors	Possible means of prevention	Killed	Injured	Deaths	Yearly total killed
							Type	Ex- tent	Limited by								
1	1839: Mar. 18	Black Heath	Chesterfield	Slope, gassy	85	60	-	Gas, Violent	Going to surface	Door left open over Sunday.	Open light.	No inspection before Monday shift.	Flame safety lamp and inspection.	40	3	-	40
2	1850	Cox's pit, Clover Hill	do	do	-	-	-	Gas	-	-	Probably open light.	-	Probably flame safety lamps and inspection.	7	-	-	7
3	1854: May 15	Chesterfield		do	-	-	20	do	-	Unventilated old workings.	Open light.	-	Ventilation and inspection, flame safety lamps.	20	0	0	20
4	1855: Mar. 19	Midlothian	-	do	-	-	-	Gas, Violent	Expansion	Blast released gas from old workings.	Blasting.	-	Ventilation, inspection, supervision, drill holes ahead.	55	5	-	-
5	1859: Apr. 13	Bright Hope	Chesterfield	do	-	-	-	Gas Local but strong	-	Open door, gas gathered in heading.	Open light.	Aided by kegs of powder.	Supervision, inspection, flame safety lamps, careful storage of explosives.	9	1	-	9
6	1863	Raccoon	do	-	-	-	-	do	-	Soldiers, stripping mine.	do	-	Ventilation, inspection, supervision.	17	-	-	17
7	1867: Apr. 3	Bright Hope	do	Shaft, gassy	-	-	69	Gas, Violent	-	Door removed.	do	Neglect of gasmen.	Supervision, ventilation, inspection.	69	0	0	69

TABLE 2.-- Gas and dust explosions in Virginia coal mines with resultant fatalities or violence,
1839 - 1933 -- Continued

Case	Date	Mine	County	Type of mine	Daily tonnage	Men Average	Explosion		Cause of accumulation	Source of ignition	Other factors	Possible means of prevention	Killed	Injured	Escaped	Yearly total killed		
							Type	Extent									Limited by	
8	1875	Raccoon pit	Chesterfield	Slope, gassy	-	-	3	Gas	Local	Small amount.	Unventilated working place. Supt. and 2 gasmen.	Open light.	Used both Davy and open lights.	Exclude open lights.	3	0	0	3
9	1876: May 20	Midlothian	-	Shaft, gassy	-	-	10	do	Violent	Expansion	Inadequate ventilation.	do	Foreman investigating reported gas with open light.	Ventilation, inspection, flame safety lamps.	8	2	0	8
10	1882: Feb. 3	do	-	do	-	-	32	Gas dust	do	-	Sealed fire area.	do	No explanation given.	Supervision and flame safety lamps.	32	0	0	32
11	1884: Mar. 13	Laurel	Tazewell	Drift, non-gassy	-	500	155	Dust	do	-	Blast blocked drifts. Main doors open.	Blasting.	Mine dry and dusty.	Safer blasting.	112	-	-	112
12	1900: May 10	Lee	Wise	do	450	100	5	do	do	Expansion	Very dusty, ineffective sprinkling; 2 rescuers killed.	do	20 sticks- open shot on rock.	Safer blasting, dust removal, watering.	7	0	0	7

TABLE 2. - Gas and dust explosions in Virginia coal mines with resultant fatalities or violence,
1879 - 1933 -- Continued

Case	Date	Mine	County	Type of mine	Daily tonnage	Average men employed	Explosion		Cause of accumulation	Source of ignition	Other factors	Possible means of prevention	Killed	Injured	Escaped	Yearly total killed
							Type	Ex- tent								
13	1901: Nov. 14	Pocahontas	Tazewell	Drift, non-gassy	-	-	-	Gas Violent	-	Fire de- ranged ventila- tion ig- nited ac- cumulated. gas.	Mine fire.	-	9	20)
13A	Nov. 22	do	do	-	-	-	8	do	-	Party went in to in- spect after ex- plosion of Nov. 14. Killed by CO.	do	Ventilation and test for CO	8	0	0	17
14	1906: Oct. 3	do	do	Drift, gassy	-	-	80	Gas, Strong dust	Expan- sion and wet sec- tions	Black blast- ing pow- der (solid)	Blown- out shot.	Shifts changing 74 plus 2 in re- covery.	36	0		36
15	1907: Mar. 16	Greeno	Wise	Slope, gassy	100	30	10	do	-	Changing from fur- nace to fan. Ven- tilation inter- rupted.	Open light.	Closed lights, super- vision, clear mine be- fore men enter, water.	6	2	2	6

TABLE 2 -- Gas and dust explosions in Virginia coal mines with resultant fatalities or violence, 1839 - 1933 -- Continued

Case	Date	Mine	County	Type of mine	Daily tonnage	Men Average employed	Explosion		Cause of accumulation	Source of ignition	Other factors	Possible means of prevention	Killed	Injured	Escaped	Yearly total killed
							Type	Ex- tent								
16	1909: Jan.	Carbon Hill	Chesterfield	Slope, gassy	-	-	-	Dust	Probably solid shooting & black blasting powder.	Blasting.	Probably tamped with coal dust.	Safer blasting, ing, water, dust removal.	6	5		6
17	1910: Dec. 14	Greeno	Wise	do	50	15	12	Gas, Strong	Gas forced down on lights by brushing with air hose.	Open light.	No State inspection service.	Closed lights ventilation, inspection, water.	8	1	3	8
18	1911: Jan. 21	Carbon Hill	Chesterfield	do	150	90	-	do	Miner crossed danger board in- to room marked off.	do	-	Closed lights, super- vision, ventila- tion.	7	5		7
19	1912: June 19	do	do	do	150	90	-	Gas	Man passed danger sign into gas in gangway face with open light.	do	-	do	2	8		
20	July 16	do	do	do	150	90	10	Gas, Small dust	Lit gas while splitting fuse with match. dust from 1st hole ignited.	Open flame	Inadequate ventila- tion, in- spection, and super- vision.	Ventilation, inspection, permissible blasting.	8	1	1	10

I.C. 6766 TABLE 2. - Gas and dust explosions in Virginia coal mines with resultant fatalities or violence,
1839 - 1933 -- Continued

Case	Date	Mine	County	Type of mine	Daily tonnage	Average men employed	Men In mine	Explosion		Cause of accumulation	Source of ignition	Other factors	Possible means of prevention	Killed	Injured	Escaped	Yearly total killed	
								Type	Ex-tent									
21	1915: Aug. 15	Parrott	Pulaski	Slope, gassy	-	-	-	Gas	Local	Small amount	Fireboss fanning out gas. Davy lamp.	Open light.	Man came into crosscut with open light.	Ventilation, supervision, closed lights.	1	0	1	1
22	1918: Jan. 5	Reed Creek	Lee	Drift, non-gassy	40	10	7	Dust	Strong	Inert dust, expansion	Solid shooting 2 or 3 blownout shots.	Blown-out shot.	Black blasting powder, "bug" dust tamp.	Cut coal, permissible blasting, water	4	0	3)
23	July 14	Roda No. 3	Wise	Drift, gassy	1,000	250	-	Gas	do	Expansion, water pool	At face in by last x-cut.	Open light.	Safety lamp on belt unlit.	Closed lights, supervision.	1	0	0)
24	1925: June 9	Linden No. 2	do	do	-	-	-	do	Local	Expansion, pillar workings	A second fall pushed gas from gob over light.	do	Some dust burned.	Ventilation, inspection, closed lights.	0	3	0)
25	1927: Mar. 24	Splashdam No. 5	Dickenson	do	500	100	10	do	Small	Expansion	Air short-circuited in starting a new wall face.	do	Men examining area.	Closed lights, ventilation, inspection.	0	8	2	0
26	1928: Aug. 25	Great Valley	Pulaski	Slope, gassy	800	70	2	do	Local	Inert dust	Fan stopped until men entered. No inspection. Acc-tive section on return.	Electric arc.	Put current on battery	Electric equipment on intake, continuous ventilation, inspection.	2	0	0	2

TABLE 2. - Gas and dust explosions in Virginia coal mines with resultant fatalities or violence,
1839 - 1937 -- Concluded

Case	Date	Mine	County	Type of mine	Daily tonnage	Men		Explosion		Cause of accumulation	Source of ignition	Other factors	Possible means of prevention	Killed	Injured	Escaped	Yearly total killed
						Average employed	In mine	Ex-tent	Limited by								
27	1932: Jan. 18	Parrott	Pulaski	Slope, gassy	320	117	18	Gas, Strong	Partly inert dust	Inadequate ventilation and inspection.	Smoking.	Propagated by gas and dust mixture.	Ventilation, inspection, supervision.	6	0	12)
28	Feb. 27	Boissevain	Tazewell	Shaft, non-gassy	2,200	300	42	Dust	Wet section, expansion	Dust raised by 4 bot-tom holes ignited by 4 over-charged top holes.	Blown-out shots.	Stemmed with bug dust.	Permissible blasting, rock-dust.	38	0	0) 54
29	June 13	Splashdam No. 6	Dickenson	Drift, gassy	300	80	10	Gas, dust	Water hole, expansion	Fan started as men went in.	Open light.	No inspection.	Ventilation, inspection, closed lights, rock-dust.	10	0	0)
Total														531	69	22	531

TABLE 3. - Gas and dust nonfatal ignitions, Virginia

Date	Field	Type of mine	Cause	Source of ignition	Injured
1850	Richmond	Gassy	Poor ventilation and inspection	Open light	1 miner
1860	do	do	do	do	do
1872	do	do	Strong feeder	do	1 super-intendent
1872	do	do	do	do	do
1883	Pocahontas	Non-gassy	Dry, dusty, black blasting powder, dust, flame, and keg of powder	Blown-out shot	2 miners
1903	Anthracite	Gassy	Clearing gas with pipe and steam jet	Open light	1 foreman
1904	do	do	Entered idle unventilated room, no inspection	do	1 foreman 1 fireboss
1910	Richmond	do	Overcharged hole in rock, black blasting powder and dust	Blown-out shot	10 men
1910	Southwest	do	Clearing gas from roof cavity with compressed air	Open light	2 miners
1914	do	do	Cut into roof feeder, miner returned after shot	do	1 miner
1916	do	do	Surveyors in old works, poor ventilation, no inspection	do	2 engineers
1916	do	do	Cut into old workings containing CH ₄ , CO ₂ , and air	do	1 miner
1916	do	do	Went over slate fall into idle working	do	1 trackman
1916	do	do	Went back into face shortly after cutting	do	2 miners
1917	Anthracite	do	Put head in hole, x-cut air course to heading	do	1 miner
1918	do	do	Cutting air course, poor ventilation, raised light too high off floor.	do	do
1918	do	do	Fanning out gas over open light	do	5 miners
1922	Southwest	do	Poorly ventilated face, faulty or no inspection	do	1 miner
1923	Anthracite	do	Fanned gas back on lamp	do	do
1923	Southwest	do	Assistant foreman passed through curtain, had safety lamp, but was smoking	Smoking	1 section foreman
1924	do	do	Small pocket. (Before they used closed lights)	Open light	1 miner
1924	do	do	Drilling in heading, poor ventilation	Electric drill	do
1924	do	do	Drilling in heading, went back too soon after shooting	do	2 miners

TABLE 3. - Gas and dust nonfatal ignitions, Virginia--Continued

Date	Field	Type of mine	Cause	Source of ignition	Injured
1924	Southwest	Gassy	Drilling in air course heading	Electric drill	1 miner
1925	do	do	Cut into old borehole	Open light	2 miners
1925	do	do	Slate fall in "robbed" area drove out gas over open light	do	3 miners
1925	Anthracite	do	Brattice shot down, not replaced until gas gathered	do	1 miner
1925	do	do	Airway blocked by coal shot down	do	do
1925	Southwest	do	Gas near roof over fall, foreman examining	do	1 foreman
1926	Anthracite	do	Examining rise room, opened safety lamp to relight	do	1 fireboss
1927	do	do	No inspection at face heading	do	1 miner
1927	Southwest	do	Loading coal	do	do
1927	do	do	Fanned gas back over lamp, door left open	do	do
1928	Anthracite	do	Examining high cavity, opened safety lamp to relight	do	1 fireboss
1928	do	do	Fixing machine, poor ventilation and inspection	Blown-out shot	4 machine-men
1930	do	do	In roof-fall cavity, faulty or no inspection	Open light	1 miner
1931	Southwest	do	Engineer went past wing curtain in air course	do	1 engineer
1931	Anthracite	do	Shooting on solid, dynamite and dust	Blown-out shot	4 miners
1932	Southwest	do	Foreman inspecting for reported gas with open light	Open light	1 foreman
1932	Anthracite	do	West up to face after shooting without extending curtain	do	2 brattice men
1933	do	do	Looked into shot holes with open light	do	1 miner
1933	do	do	do	do	do

Table 4 shows the explosions listed in table 2, arranged according to occurrence by coal fields:

TABLE 4. - Occurrence of explosions by coal fields

Field	Period covered	No. of explosions	No. killed	Average no. killed per explosion
Richmond Basin	1839-1912	14	283	20.2
Pocahontas	1884-1932	4	203	50.8
Pulaski	1915-1932	3	9	3.0
Southwest	1900-1932	8	36	4.5
Totals and averages	-	29	531	18

In the Richmond Basin the earliest reported explosion was that in 1839. Reference has been found to a possible explosion in 1817 and others in 1841 and 1844, accounts of which, however, were not discovered. The list of explosions in this area would, no doubt, have been larger had the mines not been virtually idle from about 1890 until the small revival about 1906.

The record of the Pocahontas field in Virginia is one of heavy disasters at rather long intervals. The Pulaski or semianthracite district has a light record as regards fatal explosions, probably because of the nature of the coal dust. Nonfatal ignitions have been numerous.

The record of the Southwest field indicates a more or less average performance compared with that of neighboring fields in other States.

In the 29 explosions with fatalities or violence listed for the State only 69 men escaped with injuries; these escaped death in 13 of the explosions, and in the other 16 no nonfatal injuries occurred in so far as the records indicate. As a whole, the list of those who escaped uninjured is too incomplete to permit conclusions to be drawn; however, in the last 10 years 62 men were killed in explosions in Virginia coal mines, and 18 escaped uninjured, 12 of these being in an explosion in a semianthracite mine.

Table 5 presents the fatality rates from gas and dust explosions per million man-hours of underground exposure. The rates are erratic because of the irregular nature of these disasters; they show no particular trend. The production and employment totals are relatively too small for generalization on this type of accident.

TABLE 5. - Explosion fatality rates and percentage of total, by periods

Period	Fatalities per million--			Percentage of total number killed--	
	Tons, Va.	Man-hours (inside)		Va.	U.S.
		Va.	U.S.		
1901 - 10 10 years	1.75	0.60	- -	20.9 <u>1/</u>	20.0
1911 - 20 10 years	.26	.12	0.11	4.9	12.4
1921 - 25 5 years	.00	.00	.35	.0	14.8
1926 - 30 5 years	.03	.02	.31	.9	12.5
1931 - 1 year	.00	.00	.13	.0	6.0
1932 1 year	7.02	6.00	.25	76.1	14.3
1901 - 32	- -	- -	- -	11.4	14.0

1/ 1902 - 05, estimated.

CAUSES OF EXPLOSIONS

Three fourths of the explosions with resultant fatalities or violence were charged to open lights, including 1 by a mine fire and 1 by smoking. About one fourth were caused by blasting, and 1 explosion was ignited by an electric arc.

TABLE 6. - Causes of explosions and ignitions in Virginia coal mines, 1839-1932

Explosions with resultant fatalities or violence listed in table 2			Ignitions without fatalities or violence listed in table 3		
Causes	Explosions	Percent	Causes	Ignitions	Percent
Open lights or flames	21	73	Open lights or flames	35	83
Blasting	7	24	Blown-out shots	4	10
Electric arc	1	3	Electric arc	3	7
Total	29	100	Total	42	100

Of the 42 nonfatal ignitions 35 were caused by open lights, including at least 1 by smoking. Three, caused by electric drills of a nonpermissible type, burned at least 4 men in 1 mine; these drills were subsequently discarded.

- Of the 21 fatal or violent explosions caused by open lights, the Richmond Basin, 6 in the Southwestern field, 2 in the Pocahontas mines, and 1 by a fire in the Pocahontas field. In the of the 6 explosions listed for the coal mines of Virginia open lights or smoking.

Two explosions in the Richmond Basin were charged to blasting; as field were caused by blown-out shots; and 2 in the South- charged to the same cause. In 3 cases shooting was done on back blasting powder was used; in another a heavy, open charge exploded on a large piece of rock. In most of these cases of them) the charge was stemmed with coal dust.

TYPES OF EXPLOSIONS

mines in table 2 as gassy or nongassy is based on the actual ce of gas, according to standards expressed in Mine Safety (See p. 20)

Types of explosions that have occurred in coal mines of Virginia

Explosions with resultant fatalities or violence			
Gas	Dust	Gas and dust	Total
0	4	0	4
13	1	11	25
13	5	11	29
87	167	277	531

sions described in table 2 are summarized in table 7 to show occurred in gassy and nongassy mines; they are grouped accord- f explosion, as gas, dust, or gas and dust in combination.

explosions charged to gas alone occurred in the mines of the d caused 66 of the 87 fatalities. Of the remaining cases, 1 ntas field, resulting in 17 deaths, and 2 were in the Pulaski ths. The other 3 were in mines of the Southwest district, resulting. Gas explosions caused an average of nearly 7

ons caused a heavy loss of life in nongassy mines. Two in eld resulted in 150 deaths. Two others were in the South- l the other in a gassy mine in the Richmond Basin. All 5 of ere initiated by blasting or using explosives or by prac-

The 11 explosions involving both gas and dust caused 277 fatalities, or about 25 fatalities per explosion. Six were recorded in the Richmond Basin, 3 in the Southwest, and 1 each in the Pocahontas and Pulaski fields.

PREVENTION OF EXPLOSIONS IN VIRGINIA COAL MINES

Even in the early period when frequent and disastrous explosions were occurring in the Richmond Basin, methods and means were at hand to prevent or greatly reduce the possibility of their occurrence. Customary practice and the belief that adoption of protective measures was unwarranted seem to have been the main reasons for not making use of the knowledge and means developed by past experience. On the other hand, there is no doubt that then, as now, some officials in responsible charge of the mining operations were not aware of the dangers confronting them and were shocked and surprised when disasters occurred. For at least 15 or 20 years means have been as readily available for preventing explosions in Virginia as in any other State in the Union.

Recommendations to prevent recurrence of similar explosions were made in reports on these explosions by engineers of the United States Bureau of Mines and other investigating bodies. These recommendations, representing the best accepted safety practices at the time when made, are given in table 8.

TABLE 8. - Recommendations for preventing Virginia explosions
made at time of explosions

Recommendations	Times made
Inspection for gas and ventilation of mine or section before men enter	19
Ventilation, improve or adopt proper system	17
Supervision	9
Use of permissible closed lights in place of open lights	9
Use of flame safety lamps in place of open lights	7
Dust control (watering, dust removal, rock dusting)	8
Safer blasting (eliminate overcharged holes, coal-dust stemming, solid shooting).....	4
Permissible explosives and blasting methods	3
Safer storage of black blasting powder	1

Inspection.- Failure to inspect for gas and to clear it from workings before allowing men to enter were primary factors in 19 of the explosions listed in table 2. Safety inspections, including rigid search for explosive gas, should be regularly and thoroughly made before shifts enter and at such intervals afterward as conditions may require.

Ventilation. Natural ventilation or the circulation induced through two openings or compartments in the same opening by differences of elevation or the movement of buckets or skips, or otherwise, was the chief method of

ventilation for many years. Furnaces or small blowing engines were installed to help out when more air had to be provided; later fans became common equipment. In 1914, of 102 mines inspected by the State department, 95 had fans; in 1920, 217 fans were reported in 247 mines inspected. At present it is probable that nearly all mines of any size are so equipped, although in some instances the efficiency of the control of the air may be rather low. Several important mines in Virginia have revamped their ventilating systems within the past year to obtain adequate, controlled, and more economical ventilation.

The recommendation to improve ventilation was made in 17 cases, including one in which gas was not involved, where lack of circulation of air allowed fumes and fine dust from blasting to form dangerous mixtures in the air and contributed to a dust explosion ignited by blasting.

Supervision.- Failure to exercise adequate supervision was a factor in at least 9 cases; it was the direct cause of the conditions causing some of the explosions and an indirect cause of others through failure to see that firebosses or other responsible officials were competent and diligent in performing their duties or to guard against carrying open lights into gassy workings.

The direct cause of an explosion often may be an act or accident for which those charged with supervision of the mine are not directly responsible; but they are very likely to be responsible for the conditions in the mine which permit the initiation or propagation of an explosion.

Flame safety lamps and permissible closed lights.- The Davy or the Clanny flame safety lamp was available before the earliest occurrences listed; although they gave poor light their use, at least in inspection and in workings where good ventilation was not assured, would have prevented many of the early explosions. Effective, relatively safe, closed lights were used in near-by coal-mining districts before 1910. In 1919 an installation of 125 permissible electric cap lamps was made in the Southwest field and in 1920 a second; in 1932 approximately 3,900 were in use, and this number was increased to 4,820 by the summer of 1933.

Nine explosions were caused by open lights in Virginia after practical electric cap lamps had been in use in coal mines in the United States. Twenty-eight of the ignitions listed in table 2 were caused by open lights in mines since 1910; the Bureau of Mines recommends the use of electric cap lamps of permissible type in all coal mines as a measure of safety, efficiency, and economy.

Dust control.- In 1932, 18 explosions occurred in bituminous mines in the United States. In at least 4 of these mines rock-dust localized the explosion and saved 100 to 300 lives; in 2 other mines only partly protected by rock-dusting the explosion spread to untreated workings and caused the death of 15 or 20 men.

From 1839 to the time rock-dust was proved and widely used as a protective measure, in at least 6 cases in Virginia the use of water to allay the dust and the removal of accumulations of fine coal and dust very probably would have prevented or greatly reduced the force and loss of life from explosions; in 1 instance the use of these measures did limit the spread of the explosion in a mine where several previous disasters had demonstrated the need.

As nearly as could be ascertained, only 1 mine in the State was using rock-dust before 1932. This mine was in the Southwest area. In 1932 two explosions in Virginia cost 48 lives in mines that could have been protected with rock-dust, and it is very probable that if they had been so protected few if any of the 48 lives would have been lost. Since then the mines of the Pocahontas field have been rock-dusted, and at least 3 others in the Southwest Virginia field have adopted it to some extent. Treatment of several others is understood to be under consideration.

As shown in table 9 the dust of the Richmond Basin coals was and is decidedly explosive, requiring 55 to 65 percent of incombustible matter to prevent propagation of an explosion.

No proof is needed of the explosive qualities of dust from the Pocahontas coals after actual experience. Tests in the Experimental mine on coal dust from Pocahontas no. 3 bed show that an average of 57 percent of incombustible content is necessary to prevent propagation.

The semianthracites of Pulaski and Montgomery Counties are on the border line between nonexplosive and possible explosive rank. Tests in the Experimental mine of the Bureau of Mines show that with 1 percent to 2 percent of methane in the air an explosion may be propagated by the dust of these coals.

The analyses of several beds in the Southwest field are given with their volatile ratios and the indicated percentages of incombustible material needed to prevent propagation of an explosion. This percentage ranges from 55 to 61, depending on the fineness of the dust, no gas being present.

Years of experience, full-scale tests, and studies of actual practice have caused the Bureau of Mines to recommend rock-dusting of coal mines as expressed in Mine Safety Board Decision 5.⁷

Explosives and blasting.— Permissible explosives were developed and their use advocated as a safety measure long before 1915. There were 4 explosions before that time in which use of excessive charges of black blasting powder, solid shooting, coal-dust tamping instead of clay, or firing dynamite in an open charge were the causes. In one instance gas was ignited by a match employed to light fuse, although electric cap lamps were used; this combination is by no means uncommon, although it is neither necessary nor safe.

⁷ See page 21, or Recommendations of the U. S. Bureau of Mines on Certain Questions of Safety: Bureau of Mines Inf. Circ. 6732, July 1933, 43 pp.

TABLE 9. -- Analyses of Virginia coals

Content	Richmond Basin ¹ / ₁ , average percent	Pocahontas no. 3 bed ² / ₂ , average percent	Semianthracite							
			Merimac bed		Southwest Virginia field					
			Montgomery County, percent	Pulaski County, percent	Upper Banner, percent	Dorchester, percent	Imboden, percent	Taggart, percent	Pardee, percent	
Moisture	2.0	1.7	3.6	1.4	1.0	2.8	2.6	2.4	2.0	2.8
Volatile matter	29.0	22.0	9.5	9.0	11.4	32.7	34.0	34.3	34.9	35.3
Fixed carbon	60.0	69.6	67.6	72.6	55.7	57.9	58.0	57.8	60.9	55.7
Ash	9.0	6.7	19.3	17.0	31.9	6.6	5.4	5.5	2.2	6.2
Volatile ratio	.33	.24	.12	.11	.17	.36	.37	.37	.37	.39
Indicated incombustible needed	61.0	57-58	20.0	-	40.0 ³ / ₃	55 to 61				

^{1/} Virginia Geological Survey Bull. 29, p. 108.

^{2/} Experimental mine tests.

^{3/} Indicated incombustible needed by Bureau of Mines tests with 1 percent of gas.

After permissible explosives were readily available and in more or less general use 2 explosions in Virginia were caused by the use of other than permissible explosives and permissible shooting methods.

Table 10 shows the amount of explosives of various types used in Virginia for certain years, as reported to the office of the State mine inspector.

TABLE 10. - Explosives used in Virginia coal mines

Year	Black blast- ing powder, percent	Dynamite, percent	Permis- sible, percent	Year	Black blast- ing powder, percent	Dynamite, percent	Permis- sible, percent
1913	83	13	4	1928	60	6	34
1914	78	9	13	1931	60	9	31
1920	80	5	15	1932	60	8	32
1924	67	8	25				

All forms of black blasting powder--granular, pellet, or stick--are grouped together in table 10. Until late years nearly all black blasting powder was in granular form, sold in kegs, but has been largely replaced by pellet and stick varieties. From an explosion standpoint, the hazards are essentially the same. The dynamite used is virtually entirely for shooting top rock, although types of permissible explosives are available to do this work and do it well. It is notable that there has been no appreciable change in the percentages of the various kinds of explosives used in Virginia coal mines since 1928. The hazards of shooting with nonpermissible explosives are well known, but settled practices are usually changed only when a catastrophe focuses attention on an existing hazard.

As late as 1918 an explosion was caused by shooting from the solid with black blasting powder; this practice is gradually being abandoned but has not yet been eliminated. The figures given below indicate the trend as reported by the State inspector and as given by Bureau of Mines records.

TABLE 11. - Percentage of total production machine-mined
in Virginia

Year	Percent	Year	Percent
1897	20	1915	63
1900	10	1920	70
1903	2	1925	82
1907	17	1930	86
1910	35	1931	86

These figures include both the semianthracite and bituminous production. None of the semianthracite is machine-mined, but all is shot with permissible explosives. In 1931, 88 percent of the bituminous coal was mined by machine, less than 3 percent was mined by hand, and over 9 percent was shot off the solid. In the anthracite field 22 percent was shot off the solid.

Attention is called to the following decisions of the Mine Safety Board⁸ of the United States Bureau of Mines, which refer to conditions described in this paper:

Decision 1, relating to miner's lamps in coal mines

The United States Bureau of Mines recommends:

1. In all coal mines the portable lamps for illumination be permissible, portable, electric mine lamps; and also
2. In places where fire damp or black damp is liable to be encountered, a permissible, magnetically locked, flame safety lamp for gas detection, or equivalent permissible device, be supplied to at least one experienced employee in each such place; and
3. Any employee before being supplied with a permissible flame safety lamp be examined by a competent official of the mine to assure the man's ability to detect gas; and
4. All coal mines whether classed as nongassy or gassy in any part, be supplied with magnetically locked, permissible, flame safety lamps, properly maintained and in sufficient number for all inspection purposes.

Decision 2, relating to the kind of explosives to use in coal mining

In the interest of safety the United States Bureau of Mines recommends that for blasting in coal mines, permissible explosives, fired electrically, be exclusively used; and that as an aid to blasting, all coal which is feasible to cut, should be cut or sheared.

Decision 3, defining what the Mine Safety Board considers to constitute a nongassy, a slightly gassy, and a gassy coal mine

The United States Bureau of Mines believes that all coal mines are potentially gassy; but for purposes of administration in respect to prevention of explosions and fires, the Bureau recommends the following classification:

Class 1 Coal Mine: A practically nongassy mine in which inflammable gas in excess of 0.05 percent cannot be found by systematic search.

⁸ See footnote 7.

Class 2 Coal Mine: A slightly gassy mine in which -

(a) Inflammable gas has been found* but in amount less than 2 percent in still air in any active or unsealed abandoned workings; or

(b) Inflammable gas can be found, but in amount less than 4 percent, in some place from which the ventilating current has been shut off for a period of 1 hour; or

(c) Inflammable gas can be found** but in amount less than $\frac{1}{4}$ percent, in a split*** of the ventilating current; or

(d) Inflammable gas enters a split*** of ventilating current at a rate**** of not more than 25 cubic feet per minute.

Class 3 Coal Mine: A gassy mine in which inflammable gas is found in amount greater than specified for a class 2 coal mine.

* By employing an approved flame safety lamp, with flame drawn low, or by employing an approved gas detector, or by sampling and analysis with an approved gas analytical apparatus.

** By sampling and analysis with an approved gas analytical apparatus, or by employing an approved gas detector.

*** If but one continuous ventilating current is employed in a mine this shall be considered a "split" for the purpose of this definition.

**** Determined by sampling, analysis, and ventilating-current measurement.

Decision 4, relating to auxiliary fans or blowers in coal mines and coursing of air

In the interest of safety, the United States Bureau of Mines, recommends that auxiliary fans or blowers should not be used in coal mines as a substitute for methods of regular and continuous coursing of the air to every face of the mine.

Decision 5, relating to the prevention of coal-dust explosions by rock-dusting.

To prevent the propagation of mine explosions, the United States Bureau of Mines, recommends rock-dusting all coal mines, except anthracite mines, in every part, whether in damp or dry condition. It also recommends that rock-dust barriers be used to sectionalize the mine as additional defense; but these should not be regarded as a substitute for generalized rock-dusting.

Decision 6, relating to sealing all parts of a coal mine which cannot be kept well ventilated and inspected

In the interest of safety, the United States Bureau of Mines recommends that in coal mines all entries, rooms, panels, or sections that cannot be kept well ventilated throughout or cannot be inspected regularly and thoroughly, or that are not being used for coursing the air, travel, haulage, or the extraction of coal, be sealed by strong fireproof stoppings.

Decision 7, relating to the carrying of "intake" and "return" air currents in separate shafts, slopes, or drifts

In the interest of safety, the United States Bureau of Mines recommends:

1. That the main intake and main return air currents in mines be in separate shafts, slopes, or drifts.
2. That the main intake shaft lining be of fireproof construction and there be a minimum amount of inflammable material in or adjacent to the shaft.

Decision 9, relating to the quantity and quality of air to be furnished in ventilating mines

The United States Bureau of Mines recommends in coal-mine ventilation practice the following specifications as to unit quantity and quality of air:

1. The quantity in cubic feet of pure intake air flowing per minute in any ventilating split shall be at least equal to 100 times the number of men in that split.

2. The quantity of air entering each unsealed place shall be at least 200 cubic feet per minute and as much more as may be necessary properly to dilute and carry away inflammable or harmful gases which may be present.

3. The air shall be made to circulate continuously to the face in every unsealed place into which an appreciable amount of methane enters.

4. The air in any unsealed place shall be considered unfit for men if it shall be found to contain less than 19 percent oxygen (dry basis), more than 1 percent carbon dioxide or a harmful amount of poisonous gas.

5. If the air in any unsealed place, when sampled or tested in any part of that place not nearer than 4 feet from the face and 10 inches from the roof, shall be found to contain -

- (a) More than $1\frac{1}{2}$ percent of inflammable gas, the place shall be considered to be in hazardous condition and require improved ventilation, and

- (b) If more than $2\frac{1}{2}$ percent of inflammable gas is found, the place shall be considered dangerous, and only men who have been officially designated to improve the ventilation and are properly protected shall remain in or enter said place.

6. If the air in the split which ventilates any group of workings contains more than $1\frac{1}{2}$ percent of inflammable gas, these workings shall be considered to be in a dangerous condition and only men who have been officially designated to improve the ventilation and are properly protected shall remain in or enter said workings.

Decision 11, relating to haulage and hoisting in coal mines with reference to the ventilation

In the interest of safety, the United States Bureau of Mines, recommends that in coal mines, haulage and (or) hoisting be kept in intake air as far as possible.

Decision 12, relating to methods of firing shots in coal mines

The United States Bureau of Mines extending Mine Safety Decision 2, recommends that for blasting either coal or rock in coal mines, permissible explosives or equivalent permissible device be used exclusively, and in addition recommends that in blasting -

1. Each charge shall be in a hole properly drilled and stemmed with incombustible material.

2. Each shot shall be fired separately by a permissible single-shot blasting unit, using an electric detonator or igniting equivalent of a kind specified by the Bureau for the particular permissible explosive or permissible blasting device.

3. Before and following each shot in gassy and slightly gassy coal mines, examination for gas shall be made with a permissible flame safety lamp or permissible equivalent and -

4. If more than $1\frac{1}{2}$ percent of inflammable gas is found, in the quantity and by the method specified in Mine Safety Decision 9, the place shall be considered to be in a hazardous condition and before another shot is fired the gas shall be reduced by ventilation below the percentage and quantity specified in decision 9.

5. Each shot employing explosives shall be prepared and fired by or under the immediate supervision of a man having a state certificate as a mine examiner, fire boss, or foreman; and whenever conditions permit all other men than those authorized to prepare and fire shots shall be out of the mine when shot firing with explosives is being done.

Decision 13, relating to electrical equipment in coal mines which may become gassy

The United States Bureau of Mines recommends that when electricity is used in coal mines rated as gassy, or whenever in any mine the atmosphere may become gassy:

1. Electrical equipment shall be permissible,
2. Nonpermissible electrical equipment*** shall be used only in pure intake air,
3. Electrical power shall be cut off whenever the air in the workings is in a dangerous condition, due to inflammable gas.

Decision 15, wetting machine cuttings, the face, and tops of cars in coal mining

In the interest of safety in coal mining, the United States Bureau of Mines to lessen the coal-dust explosion hazard recommends that:

1. Machine coal cuttings be wet as the cutting is being done.
2. The coal face, and the working place 40 feet therefrom, shall be kept free of coal-dust by the use of water.
3. The top of loaded cars in the working place shall be wet.

Decision 16, machine cuttings to be removed in coal mining

In the interest of safety in coal mining, the United States Bureau of Mines recommends that:

1. Machine cuttings be removed from the cut.
2. If the machine cuttings are of a character which would contribute to a dust explosion, they shall be sent out of the mine.

Decision 17, lessening the formation and distribution of coal-dust in haulageways in coal mines

To lessen the formation and distribution of coal-dust in haulage-ways, the United States Bureau of Mines, recommends that in bituminous and lignite coal mines:

1. The mine cars should be constructed and maintained dust-tight.
2. The coal should be so loaded that it will not shake off in haulage.
3. The cars and loads should be so sprayed as to prevent dust being distributed along the haulageways.

Decision 18, superintendents and foremen of coal mines should have State certificates of competency

In the interest of safety in coal mining, the United States Bureau of Mines recommends that:

1. The foreman regularly in charge of underground operations and also any person who, in the absence of the foreman, may be placed in temporary charge should each have a certificate of competency from the State to act as mine foreman.
2. The superintendent or person in responsible charge of the mine, to whom the mine foreman reports, should have a certificate of competency from the State which should be issued upon a showing of underground experience for a period of time as long as that required for a foreman's certificate and upon passing an examination including all technical questions asked in the examination required of foremen.
3. These certificates should expire after some stated period of time, such as 5 years, and should be renewed only after the applicant has again passed the examination required by the State.

CONCLUSIONS

Dust of the semianthracites has been found to be explosive in the presence of gas by both experience and tests made in the Experimental mine to determine limiting conditions. The dust of coals of southwest Virginia is by analysis clearly within the explosive range, and this has been shown by explosions propagated by dust in several mines.

Although the mines in the Pocahontas field had worked 25 years without an explosion, later experience proved that explosions can and will occur unless reasonable precautionary measures are taken, and now it should be evident that the best protection against widespread propagation of an explosion can be afforded only by adequate rock-dusting supplementing other safety features, such as effective ventilation, adequate supervision, and the use of up-to-date and readily available safe methods and equipment. The information necessary for proper application of rock-dust to these mines was obtained by means of large-scale tests conducted in the Experimental mine of the United States Bureau of Mines at Bruceton, Pa., using dust from these coals.

UNITED STATES BUREAU OF MINES
SCOTT TURNER, DIRECTOR

INFORMATION CIRCULAR

USE OF AIRPLANES IN
MINING AND PETROLEUM OPERATIONS
(ABSTRACT)



FROM AN UNPUBLISHED BULLETIN MANUSCRIPT

BY

HUGH M. WOLFLIN

(ABSTRACTED BY CHARLES WILL WRIGHT)

INFORMATION CIRCULAR

UNITED STATES BUREAU OF MINES

USE OF AIRPLANES IN MINING AND PETROLEUM OPERATIONS ¹

(Abstract)

From an unpublished bulletin manuscript

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¹ The Bureau of Mines will welcome reprinting of this paper, provided the following footnote acknowledgment is used: "Reprinted from U. S. Bureau of Mines Information Circular 6767."

² Formerly Supervising Engineer, Intermountain Experiment Station, Salt Lake City, Utah, U. S. Bureau of Mines.

³ Chief Engineer, Mining Division, U. S. Bureau of Mines.

INTRODUCTION

This Information Circular is an abstract of a manuscript which was completed in June 1933, by Mr. Hugh M. Wolflin, formerly of the U. S. Bureau of Mines. This abstract has been prepared by Charles Will Wright for presentation at the February meeting of the American Institute of Mining and Metallurgical Engineers in New York. It has been necessary to omit many of the details and most of the cost data and tables, as well as the illustrations in the original manuscript. As the use of airplanes in mining developments is making such progress and as a publication on this subject to be of greatest value should include the most recent information, it is unfortunate that, due to a lack of printing funds, it has not been possible to publish this manuscript.

The purposes of Mr. Wolflin's paper are:

- (1) To describe briefly what has been accomplished in some outstanding examples by the use of aircraft in the mining and petroleum industries,
- (2) To record costs,
- (3) To present available information and suggestions to assist in solving future air-transportation problems in the industries under consideration.

The manuscript is divided into seven chapters:

- I. Prospecting and Exploration
- II. Aerial Mapping
- III. Air Transportation of Freight
- IV. Individual Air Transportation
- V. Landing Fields
- VI. Dependability of Airplanes
- VII. General Summary

It is well illustrated by diagrams and photographs and includes a bibliography of published articles on aviation from which part of the information in the manuscript was taken. Much of the data, however, was derived from correspondence with engineers and managers of air-transportation companies and others who contributed generously such information as they had available.

In the introduction Mr. Wolflin mentions the various means of transport used in isolated regions throughout the world; trains of native packers in Africa, dog teams in Alaska, canoes in Canada, and pack horses in the Rockies. He tells how airplanes were used at first for carrying prospecting parties, engineers and geologists to investigate new districts, then for mapping the topography and geology in these outlying areas, and describes the development of the big cargo plane for transporting heavy loads to otherwise almost inaccessible regions.

PROSPECTING, EXPLORATION AND SURVEYING BY AIRPLANE

In northern Canada and Alaska, airplanes have been used for prospecting and exploration, perhaps to a greater extent than elsewhere in the World, although several extensive surveys and exploration trips have been made with them in central Australia. The substitution of airplanes for more primitive methods of transportation in the North has been considerably advanced by the development of satisfactory landing gear; skis in winter and floats in summer. Much of the northern terrain has many lakes and rivers which serve as satisfactory landing places for pontoon-equipped aircraft during the summer. Good landing places are even more numerous for ski-equipped craft during the winter. As a rule there is not more than a month or two during the spring break-up and the autumn freeze when the use of aircraft is impracticable in these areas.

The satisfactory development of air-cooled engines for airplanes has been an important factor in increasing air transportation, particularly in areas where extremely low temperatures may be encountered. With the rapid increase in dependability and horsepower developed per unit of engine weight, the cruising range and pay load of aircraft have increased, also the efficiency and safety of air transportation in remote areas.

In general, it is conceded that the use of airplanes has increased both the safety and efficiency of prospecting and exploration in areas remote from centers of population.

A number of actual examples of aerial prospecting and exploring operations are summarized here for the information of organizations considering the examination of areas remote from transportation. While the details, particularly the cost data, are fragmentary, all the readily available information has been included.

Outstanding Examples

The Consolidated Mining and Smelting Co. of Canada, with headquarters at Trail, British Columbia, has perhaps made as extensive use of airplanes in connection with its recent exploration work as any mining concern on the American Continent.

In the spring of 1929 the company brought some of its examining mining engineers to Trail and provided them with flight instruction until they received their commercial air-pilot licenses. When the ice broke on the lakes and rivers in the northern mining areas, two of these engineers started out in De Havilland Gipsy Moth airplanes, equipped with pontoons. These ships are light, two-place, open-cockpit, folding-wing biplanes with four-cylinder, inline, vertical air-cooled, direct-drive, 95-hp. Gipsy engines. Each examining engineer-pilot was accompanied by an airplane mechanic who also assisted with mine sampling and similar work.

The following year, 1930, the company trained several more engineers during the winter months. In the spring, two additional Gipsy Moth airplanes were purchased and sent out with engineer-pilots and airplane mechanics.

As aerial exploration developed successfully, three larger air-craft were operated for about 2-1/2 years. These heavier machines consisted of one Fokker Super-Universal cabin monoplane and two Fairchild (71) cabin monoplanes, each powered with a Wasp 420-hp., Pratt and Whitney, radial air-cooled, direct-drive engine. The larger planes served as freighters to carry food, gasoline, and supplies to caches near where the smaller planes were operating. They also were used to carry men and supplies, including drilling and camp equipment, into remote regions. In this way, properties under option were speedily investigated and drilled. Then smaller planes were also purchased for the transportation of the company's officials.

To March, 1931, this company's airplanes had flown approximately 425,000 miles since the first machine was put into operation. There was not a single failure of an engine in flight or a single injury of a serious nature to personnel. A considerable portion of the mileage is credited to the engineer-pilots who received their training in company-owned airplanes. Most of the flying was done under the wide variety of conditions encountered in the remote mining districts of northern Canada. In the summer time the airplanes were equipped with pontoons. In the winter time the pontoons were replaced by skis.

During 1931 "Consolidated" aircraft flew a total of 3,028.8 hours, covered 273,726 miles, and carried 1,817 passengers and 247,897 pounds of freight.

Costs of operating the two-passenger Gipsy Moth airplanes in the Canadian Northwest have been announced as approximately 35 cents per plane-mile. This figure includes all expenses such as engine and aircraft overhauls, insurance, depreciation, gasoline, oil, and all operating and administrative costs other than engineer-pilots' salaries. Overhauls and mechanical work of all kinds were done at the company's shops at Trail.

From the foregoing record it appears that mining and petroleum companies operating in the United States could reduce costs and increase supervisory efficiency considerably by giving pilot training to some of their district managers, superintendents, geologists, engineers, and highly paid service men who now spend a considerable part of their time driving automobiles. The furnishing of light, stable, slow-landing airplanes to some of these men who are building up automobile mileage at a surprising rate is recommended for consideration. From available information it appears that the operating costs of many of their automobiles are slightly higher than would be the cost of operating suitable airplanes.

A letter from John C. Rogers, consulting mining geologist, of Toronto, states that his organization has had excellent results in the operation of airplanes by mining engineers and geologists with pilot-training. He considers it advisable to have a pilot who understands thoroughly what is sought by the field men and that in isolated districts the second man should be a first-class mechanic as well as an observer or mapper. Pilots should not be permitted to fly alone.

Prior to March 1929 the Northern Aerial Minerals Exploration, Ltd., commonly designated as N.A.M.E., had flown more than 100,000 miles over northern Canada without a single casualty, according to details published by its staff in the Canadian Mining and Metallurgical Bulletin for March 1929, summarized as follows:

"Much of the flying was over virgin territory probably never before visited by white men. Some of it was over regions that abound with possibilities for minerals. All of it was done for exploration and the establishment of bases and supply stations for carrying men and their tools and provisions to remote locations."

Some advantages gained by the use of the airplane in the North are pointed out by J. E. Hammell, president of the organization, in the Transactions of the Canadian Institute of Mining and Metallurgy for 1929, as follows:

"1. Safety: From 1926 until March 1929 there were no casualties among air passengers, where about 42 men lost their lives through the hazards of more primitive types of transportation.

"2. Communication and supervision. These provide the key to effective prospecting. Parties are at all times in touch with district engineers at headquarters and receive help and advice on the ground when they need it. Effectiveness is tremendously increased.

"3. Morale. This is increased out of all proportion to other factors. Men are in touch with civilization and have had the worry and labor of transportation of supplies removed. They can apply themselves fully to the work of prospecting within a convenient radius of their main camp. When they have finished with a particular locality they, their canoes, and their supplies are transported to another area and work resumed in a fraction of a day.

"4. Aerial observation. Geological boundaries can be outlined roughly by an engineer or an experienced prospector. Topographic features of the country can be supplied to the field parties in remarkably quick time."

Transportation is probably the biggest problem the prospector has to face. According to the staff of the N.A.M.E., the airplane eliminates this for him. The prospector can be carried in a few hours to regions that ordinarily could not be reached in weeks of travel. He can be kept supplied with food, explosives, and tools.

Transportation is at present by far the most important role of the airplane. Under it may be included:

(a) Placing parties completely equipped in previously selected localities.

(b) Keeping these parties supplied with food, dynamite, drill steel, and tools. Because of transportation difficulties prospectors rarely have carried the latter aids into the wilderness.

(c) Carrying in additional men, equipment, and supplies without loss of time if results of preliminary work warrant more intensive prospecting.

(d) Moving parties without loss of time from one locality to another when the work in an area is completed.

(e) Placing parties early in the season (April) in selected localities, giving them a 6 or 8 weeks' start before the best prospecting months of the year, May and June. The difficulties in connection with placing parties before break-up by ordinary transportation methods are so great that it is rarely attempted except in localities close to railways.

By such methods one scouting party in a single season can line up areas for intensive prospecting during the following season that could easily keep a small army of prospectors busy.

Methods of operation, organization, personnel, division of personnel, lists of equipment, and details of work performed in specific areas are included in the original article cited, with a general discussion of cost factors and remarks on increase in safety due to the use of airplanes.

With the organization of flying companies in northern Canada, some of the mining exploration companies, including N.A.M.E., have found it more economical recently to buy aerial transportation than to maintain their own flying equipment and staff. When an area has been partly developed so that the addition of larger crews is justified and the amount of freight for continuous movement has increased a mining company perhaps may be justified even under present conditions in considering the air transportation of its own freight.

Views of Experienced Pilots

The prominence of Major T. G. Lanphier, president of the Bird Aircraft Corporation, in the field of aviation makes his views of special interest. The following has been summarized from his letter of October 27, 1931:

"That suitable planes can be operated safely by men of limited experience is shown by the fact that a large number of plane owners are private people who have no interest in commercial aviation and who do not fly frequently. It is possible for a normal person to learn to fly a Bird plane in 4 hours of instruction and immediately thereafter be his own observer if he is a geologist.

"A pilot with any degree of skill can land and take off safely at high altitudes with suitable planes, particularly if they are equipped with superchargers. Handling a plane in stormy weather on the ground

is simple. It is only necessary to stake down the airplane and keep it in a sheltered spot. Even if the spot is not particularly sheltered from high winds, a plane staked down properly should give no trouble.

"Cruising near the ground to examine its geological features is particularly simple with our planes. The practice is referred to as hedge hopping and it is necessary for the plane to maintain a low forward speed if the geological observations are to be accurate. It is possible to cruise along with the plane at much less than 60 miles an hour, which, from reasonable altitude, makes preliminary geological observations practical."

Newton Wakefield, vice president and general manager, Mamer Air Transport, gives some of his views in a paper on "Mine Development by Airplane" in the Engineering and Mining Journal for May 1932, page 264, as follows:

"The usefulness of the airplane in the mining industry has not been adequately appreciated or exploited. During recent years the airplane has been utilized extensively to aid mining engineers and geologists in scientifically prospecting for new orebodies and new oil deposits. This use has been rather more extensive in Canada and Alaska than in the United States.

"Usually, the plan is to fly a geologist over the region to be prospected and to let him study the general relation of outcrops and other geological features visible from the air. Aerial pictures may be taken and any interesting country mapped by building up a mosaic of aerial photographs. The geologist finally reduces the entire area under consideration to small sections which appear to have the most promise."

Central landing fields are then located, and prospectors, equipment, and supplies are flown to these points. The prospectors are assigned definite areas to cover, and every 2 or 3 weeks they return to the landing fields with samples they have taken. Samples are examined and assayed with a portable outfit set up at the landing field or are flown to the nearest assay office. The engineers and field geologists are thus able to keep complete records of the work being done. They can concentrate the prospectors in areas which give the most promise as shown by samples and examination of the work done. In this way all work is under the control and supervision of the field engineers or geologists; plans are carried out in a scientific and systematic manner; and large areas can be prospected thoroughly in the shortest time possible.

Aerial maps are complete in every detail and give the engineer a pictorial representation impossible to obtain in any other way. They contain such a wealth of detail that it is often possible to lay out roads, power lines, tramways, and other surface developments on the aerial maps without making further surveys. Aerial pictures, aerial maps, and aerial reconnaissance can help the mining engineer in hundreds of ways. In the future, the aerial camera will be as much a part of his equipment as the transit is today.

Wakefield's article describes the modern transport plane, its capabilities, and its limitations and shows how it may be employed by the mining industry in solving problems of preliminary exploration and transportation. It is stated that a transport plane of one of the earlier Ford Trimotor types can carry a 3,000-pound cargo at 100 miles an hour and that with this load it will land on and take off from small fields.

Usually, suitable landing fields can be built at small expense by felling trees and grading runways or by smoothing out the rough spots in a meadow. In the Idaho-Montana area the United States Forest Service estimates the cost of preparing this type of field as about the same as that of building a mile of trail. No subsequent maintenance is necessary in many areas.

The direct cost of air transportation of equipment into remote areas frequently is far below that of any other method. The saving in time usually means a heavy reduction in ultimate costs.

Some Unit Costs in Alaska

Frank Dorbandt, commercial airplane operator at Anchorage, Alaska, contributed a letter containing some interesting unit costs of airplane operations in his district. His observations are summarized as follows:

"Air-transportation costs vary with the size of the organization. I am operating alone and have only one mechanic. I am using a six-passenger Pilgrim monoplane with a pay load of 1,257 pounds on skis and 896 pounds on pontoons. These pay loads are set by the Aeronautics Branch of the United States Department of Commerce.

"In summer, of course, our operating costs are higher because of the lower pay loads. My operating costs average \$60 an hour, including depreciation, insurance, mechanic's salary and expenses, my own salary as a pilot, and the various other items which make up the cost of operation. Gasoline averages 50 cents a gallon and oil \$1.25 a gallon.

"I average about 600 hours a year at a cruising speed of approximately 100 miles an hour. My plane is insured for \$11,000 against fire only, at a cost of 5 percent. I carry \$10,000 to \$50,000 passenger liability and \$20,000 public liability insurance. Our insurance premiums amount to \$1,085 per year. We write our ships off in 3 years, the depreciation being 40 percent the first year and 30 percent each the second and third years.

"Our average charge per passenger-mile is about 35 cents, and our average freight charge for 200-mile distances is 30 cents per pound, corresponding to about \$3.00 per ton-mile."

Arctic and Winter Flying

W. J. McDonough, in an article published in Aviation Engineering for March 1933, stated that the chief value of the airplane as an aid to mineral exploration could be divided into eight distinct functions:

"1. The rapid and safe transportation of prospectors, food supplies, and equipment from one northern point to another, thus more than doubling the efficiency of the prospector by saving time.

"2. The extension of the prospecting season by many weeks because of the ability to reach quickly remote districts in a few hours.

"3. Opportunity to visit prospecting parties by regular patrol and supply their requirements as well as keep them in touch with headquarters and to fly them out in case of business demands or illness.

"4. Coordination of a company's exploration effort by flying supervisory engineers and geologists to any district from which important information has been received through various bases.

"5. The rapid inspection of claims regarding which information has been received and the confirmation or rejection of reports which otherwise might cause unwarranted effort.

"6. Flights by large freight aircraft carrying complete mining staff for the investigation of distant areas, with supplies and equipment sufficient for extensive examination of properties.

"7. Flights carrying directorial heads on tours of inspection to ascertain water-power resources and timber supply of any district in relation to mining development.

"8. Cooperation with Government departments in the rapid organized development of northern Canada as regards its mineral resources."

High-wing monoplanes are preferable, due to conditions of operation encountered in the field. It was stated that nearly all airplanes used on transportation service in Canada have been equipped with Pratt and Whitney Wasp 9-cylinder, radial air-cooled engines. Strong preference was expressed for single-engined planes because of the delay due to warming engines with plumbers' pots to get them started in the morning. A cruising speed of 100 miles per hour for 6 hours without refueling is considered desirable, but it is extremely important that landing speed under full load should not exceed 53 miles per hour. The airplanes most suitable for use should be able to take off from glassy water with a pay-load of 2,300 pounds and 5 hours' fuel supply in 40 seconds. Desirable minimum pay-load capacity of cabins was given as 250 cubic feet, although planes in use were said to have only 150 to 170 cubic feet of pay-load space.

In conclusion, it was stated that take-off ability and low-landing speed should be given the primary consideration for this type of service. The application of slots and flaps seems to be a solution to this problem. A landing speed of 60 to 65 miles an hour is much too high for safe operation and makes conditions extremely dangerous when a landing becomes necessary on rough water or on hard-packed or drifted snow.

H. A. Oaks on Air-Transport in Canada

H. A. Oaks, Airways, Ltd., Toronto, in a letter to the author makes the following statements in reference to planes for transport:

(a) Transport aircraft to have sufficient capacity to be effective must have at least 400 hp. An air-cooled engine of proved type is preferable. Reconnaissance aircraft should have air-cooled engines of 150 to 200 hp.

(b) Disposable load (total licensed load less weight of aircraft empty) should be about 2,700 pounds for transport and 1,200 pounds for reconnaissance.

(c) All-metal construction is preferable. Even if a hangar is available at headquarters, the airplanes will be in the open most of the time.

(d) Cruising speed with floats should not be less than 95 miles an hour. It is advisable to have transport and reconnaissance machines with approximately the same cruising speed so that they will be able to travel together conveniently.

(e) Rugged construction for possible winter operation in the Far North, with consequent rough going.

(f) The aircraft should be seaworthy to give maximum service for extended summer operations, some of which may be on big inland waters. Delays due to weak float undercarriage are expensive. Unnecessary risks of capsizing or foundering cannot be tolerated.

(g) Quick take-off and good climbing ability are highly essential for operation into and out of small lakes and rivers. The equipment should have a service ceiling of at least 15,000 feet to insure satisfactory operation in the mountains.

(h) Adaptability for aerial photography is required.

Personnel. -- To save weight and expense men of dual capacity should be chosen. Pilot-mechanics are obtainable. They can be trained to assist in mine sampling. Pilot-geologists and pilot-engineers are rare but are extremely desirable for such work. Experience has shown that an engineer or geologist who has not flown considerably as a pilot is a distinct financial hazard. Too frequently, he loses his effectiveness as a geologist and engineer if his flight training and experience are inadequate.

"Authority to use aircraft. -- To eliminate joy-riding and unnecessary flying, all flying is classified as "reconnaissance," "transport," "test," or similar service, and each flying job must be requisitioned by someone in authority."

After the cost of flying is estimated, a rate should be set up, making a fixed charge per mile or per hour or per pound, depending upon the nature of the work. All flying should be charged at this rate against the operation for which it was performed. It is advisable to make the rates high enough to give the flying branch a small paper profit on its operations.

A complete field record of flying operations should be kept.

Aerial Oil Reconnaissance

Following are fragmentary notes from correspondence and conversation with Stuart Moir, western sales manager for Fairchild Aerial Surveys, Inc.:

"Some emphasis should be placed on the fundamental difference in the requirements for surveys made for petroleum reconnaissance as against those made to locate mineral deposits. Generally, the following has held true for experience and practice in the United States:

1. Petroleum surveys have covered large areas and required relatively small-scale maps.
2. Mineral surveys usually have covered relatively small areas and required large-scale maps.

The big area in Africa mapped by the Aircraft Operating Co. of London covered over 60,000 square miles, according to available information. These surveys were for a three-fold purpose: (1) Land classification, (2) locating mineral-bearing areas, and (3) mapping a new country prior to colonization.

From the Dallas, Texas, headquarters aerial maps were prepared of more than 30,000 square miles for oil reconnaissance. The area included many of the major producing fields, such as Yates Pool, Winkler, Luling-Mexia fault line and the Balcones fault line in Texas, and the entire East Texas Basin. Hobbs field in New Mexico was completely mapped as were many of the domes in the Gulf region of Louisiana and Texas.

In Mexico extensive surveys have been made in the States of Tamaulipas, Nuevo Leon, Vera Cruz, Tabasco, and Yucatan.

Considerable visual reconnaissance by airplane has been done by mining engineers and geologists in Sonora, Chihuahua, Sinaloa, Zacatecas, Durango, and Guerro, and there are probably other aerial surveys of which nothing is generally known. The Military Engineer for March and April, 1933, has details of the Guatemala-Honduras boundary survey, made through cooperation of the United States Geological Survey and the United States Army Air Corps.

The cost of operating a four-passenger cabin Fairchild airplane (FC2) in Mexico over a period of approximately 4 years was 22.75 cents a mile.

This included depreciation, taxes, and all operating expenses. All charges for the operation of an EC2W Wasp-powered Fairchild in Mexico came to 28 cents a mile.

The operation of a seven-passenger Fairchild 71 with a Wasp engine amounted to 30 cents a mile over 700 to 750 hours of flying.

Exclusive of the pilots' salaries, the charge for operation of a 300-hp. photographic plane is about \$30 an hour, based upon about 400 hours of flying per year.

Flying for photographic work sells for \$50 to \$60 an hour, including the pilot and taking into account the relatively small number of hours flown in a year. This figure includes \$2.71 an hour for insurance of the type carried for photographic work (\$100,000 to \$200,000 public liability, plus coverage for plane damage by tornado or fire, plus theft protection).

The cost of aerial maps and surveys depends upon so many factors that an offhand estimate is likely to have little value. Such factors as the scale of the maps and the probable use to which they will be put are important. The elevation of the area to be mapped is of particular importance. Distance from headquarters of the mapping concern and whether there are other jobs to be done in the immediate future in the area where the mapping is undertaken, have a decided bearing on costs. The safest way to arrive at the cost of the work is to prepare specifications covering the scale and the accuracy required, and other details, and to call for estimates for concerns accustomed to making aerial maps.

In making oil surveys, we are accustomed to think in terms of square miles, whereas in making mineral surveys usually we think in terms of acres.

Aerial-map interpretation is an art. A high degree of efficiency in reading this type of map can be acquired only after considerable study and field experience.

There is every indication that with the rapid progress in aerial development, costs of aerial maps will be reduced and their accuracy increased in the near future.

Among the advantages of using aerial-survey methods for pipe-line location may be mentioned the following:

1. A right-of-way man can show a land owner exactly where the proposed line will cross his ground, without the owner being antagonized by surveyors running lines on his property; also, a survey can be made without the land owner realizing it.

2. The shortest and most convenient and hence the most economical location can be made within a minimum time.

3. Changes in operating plans which may require changes in location occasion a minimum of lost time for resurvey. Due to the large amount of detail on aerial-survey maps, changes can be considered and decisions reached immediately in many cases.

4. The pipe-line contractor can submit a more intelligent bid owing to the large amount of detail shown on the aerial-survey map.

5. Transportation and stringing costs are reduced because trails, roads, highways, and topography near the right of way are shown in great detail.

6. Obstacles are shown in such detail that it is easy to avoid them.

7. Stream-crossing expense is minimized by selection of the more favorable crossings. The wealth of detail on the photographs makes it possible to estimate stream-crossing costs more accurately.

Aerial Surveys in Relation to Economic Geology

Bulletin 337 of the Institute of Mining and Metallurgy, October, 1932, contained an interesting paper by Donald Gill on "Aerial Survey in Relation to Economic Geology." The following points of practical interest to the geologist are given special mention:

1. It is desirable that all the photographs in a mosaic should have been taken at approximately the same time of day to avoid confusion in shadow effects.

2. It is highly desirable that all joins should be straight edges, rather than that they should be camouflaged by following some undulating feature, for example, a road, the edge of a forest, or a stream, as is sometimes done. It should be impossible to mistake a join for anything else in mosaics used for scientific purposes.

3. It is preferable that there should be a frank break in detail at a join rather than that the mapping should be fudged in any way - for example, by stretching the emulsion of the prints when wet.

The proper use of aerial surveys and photographic maps expedites and increases the accuracy and detail of geological and other work in connection with the development of mineral deposits. The savings effected in the development and other work usually will exceed considerably the cost of the aerial work. Furthermore, the right kind of aerial surveying will minimize the chances of overlooking nearby deposits which may prove to be of economic value.

Aerial surveys expedite and reduce the cost of construction of railroads, highways, trails, aerial tramways, power transmission and telephone lines, pipe lines, and similar auxiliary developments.

AIR TRANSPORTATION OF FREIGHT

Recent developments indicate that where the cost of ground transportation of freight exceeds 11.5 cents per ton-mile consideration should be given to the use of aerial transportation.

Where the air transportation of all freight is not justified the employment of auxiliary airplanes is suggested for consideration to reduce expensive delays in the shipment of bullion, high-grade concentrates, perishables, emergency repair parts, emergency supplies, samples, drill cores, and similar items.

In the following summaries the reader should bear in mind that many of the instances cited were pioneering efforts. The experience gained may indicate the advisability of variations in methods and equipment on future transportation jobs. Development of both equipment and methods has been extremely rapid. Hence, yesterday's best practice in air transportation may be poor when measured by the standards of tomorrow.

Without defending or criticizing the types of equipment chosen, the author has given such available cost data and details as could be procured for some actual uses of aircraft to transport mining and petroleum equipment and supplies into isolated districts.

Freight Transportation in New Guinea

One of the outstanding accomplishments is described by C. A. Banks, managing director, Bulolo Gold Dredging, Ltd., in an article published in Bulletin 334, July, 1932, of the Institution of Mining and Metallurgy, London. This is supplemented by additional data from the Junkers Corporation of America in July, 1933.

The Bulolo placer deposits are situated in New Guinea approximately 70° south of the equator and 35 miles in a direct line from the coast. The surface elevation averages 2,250 feet above sea level. As there are no roads through the dense tropical jungle, drilling and shaft testing were conducted with supplies and equipment sent from the coast to the interior by airplanes capable of transporting about 2,000 pounds per trip.

When sampling results had proved satisfactory it was decided to equip the property with two 10-cubic foot, all-steel dredges (now working) and a 3,000-hp. hydroelectric plant, and possible methods of transporting such equipment were given full consideration.

The country between the coast and the dredging ground is rough. A mountain range, the lowest pass through which is about 4,000 feet above sea level, has to be crossed. A road would have to be 90 miles in length. Construction would require 12 to 18 months. It would cost more than a million dollars. Even with a road completed, the cost of hauling would be heavy. Road upkeep would be extremely expensive because of the heavy rainfalls.

Air transportation of men and supplies for preliminary testing and prospecting operations had been entirely satisfactory. After full consideration of the many factors involved, including sectionalizing of equipment to reduce the size and weight of individual pieces, it was decided to abandon all idea of a road and to transport all machinery and equipment from the coast by airplanes. The total weight of the two dredges and the hydro-electric plant was estimated at 2,400 short tons. It would be necessary to transport 200 tons monthly not to delay construction.

Transportation equipment. - Two Junkers C-31 trimotored, low-wing passenger monoplanes, altered and strengthened to suit requirements, were purchased. With any two of the engines in operation the maintenance of a reasonable elevation is possible with these planes.

Spare parts. - During the transportation of 2,500 short tons, requiring about 1,000 trips, engine trouble developed in one flight only, when the center engine failed. The loaded plane finished the flight without difficulty with the remaining two engines.

Landing fields. - Construction of a suitable landing field at Bulolo was a major undertaking. A sandy gravel site was chosen on the downstream end of the property just off the dredging area. Dense, tropical jungle with trees exceeding 4 feet in diameter had to be cleared and a large drain placed along the upper edge to keep the field free of surface water. To reduce bumpy landings due to air currents produced by reflected heat of the sun, the entire area was planted with couch grass.

The Bulolo air field took 7 months to complete and cost approximately \$30,000. Cutting the grass has been the only maintenance required since completion. With planes weighing approximately 19,000 pounds striking the ground at 75 miles an hour a soft surface cannot be tolerated. With good surfacing the wear and tear from landings and take-offs has been small.

Airplane operations. - Airplane operations were conducted by an airways company under a contract which included management and other facilities.

The flying distance from the coast airport to Bulolo and return is approximately 100 miles. The flying time for the round trip averages 75 minutes. The average altitude attained in crossing the mountains is 6,000 feet. Loads carried have averaged about 5,400 pounds. With improvements in the Lae landing field, however, they will be increased to about 6,000 pounds or more.

The airplanes are designed to carry long pieces as well as heavy pieces, hence they cannot be flown empty. For the return flight ballast must be placed in the aft compartment of the otherwise empty plane to make the tail weight approximately 2,000 pounds. Water ballast could be used, but gravel was needed at Lae, hence gravel in bags was used for ballast.

Loading and unloading operations. - A 10-ton, locomotive-type crane loads the planes at Lae. Its position is just behind the wing. Cargo is always loaded and unloaded from the rear so that in case of accident the load would fall on that part of the fuselage which is most easily repaired.

The concrete loading apron is fitted with platform scales for weighing the tail of the plane to make sure that its weight always is between 1,800 and 2,000 pounds when the loading operation is completed. Cargo is shifted to keep the weight between these limits.

After a little experience, loading time was reduced to 20 minutes or less. Only ten minutes were required to load dredge buckets.

A careful inventory of each load was made to facilitate replacement with minimum delay in case of accident.

A hand-operated, stiff-leg crane is used for unloading at Bulolo. Unloading requires about half the time needed for loading.

The anticipated difficulty in securing cargo in the cabin did not materialize. Heavy pieces of machinery were lashed in position by light, steel cables attached to the floor and sides of the cabin. The cables were fastened around the machinery with clamps and drawn tight with turnbuckles. No trouble has been experienced due to cargo shifting.

The upper tumbler shaft, which was 12 feet long and weighed nearly 7,000 pounds, was carried in four cradles set on the main transverse beams of the cabin floor. With concentrated loads, rubber pads were introduced between the load and the wooden cradle to absorb shock in landing and taxiing. Some less heavy, but awkward, parts were carried on bags of rice needed for food.

Progress. - Two additional dredges and two more hydroelectric units are in course of transportation. One of these additional dredges was scheduled for operation in September, 1933 and the next in May, 1934. In addition, general supplies, spare parts, and other necessities are being transported continuously for the operation, bringing the weight of material transported by airplane well above 5,000 tons.

Costs. - Total field costs for air transportation of 2,500 tons, including all handling of cargo from ship side to airplanes, were \$182,000, or \$72.80 per short ton, equivalent to \$1.45 per ton-mile (using \$3.50 to the Australian pound).

The records to July, 1933, show that the airplane service for the Bulolo Gold Dredging, Ltd., includes 12,500,000 pounds of freight, 11,000 passengers over a period of 5 years, and a total distance traveled of 1,000,000 miles. The staff included 7 pilots, 20 mechanics, and 28 helpers. This service has been carried on without the loss of any freight and without a serious accident.

Freight Transportation By Alaskan Airways

Following is a brief of a statement by K. K. Katz of the Alaskan Airways, Inc:

Air transportation is already an important factor in Alaska. Considering the sparsely settled nature of the country, doubtless it will become increasingly important in the future.

Instances where aircraft are used in the development of mining operations in the Territory are of almost daily occurrence. Radio communication is fairly well established throughout the interior and requests are received regularly at the bases of the Alaskan Airways for replacements of broken machinery for spare parts, and supplies.

An outstanding instance of the use of the airplane in this connection occurred in the spring of 1932: Sixty or more bucket lips, weighing approximately 100 pounds each, were taken by airplanes from Anchorage to the dredge of the New York-Alaska Gold Dredging Co., 500 miles or more in the interior, in the vicinity of Akiak. The only other way to have transported these lips would have been by steamer from Seattle to Bethel at the mouth of the Kuskokwim River, where they would have been transhipped to a small river boat and carried some distance farther. They would then have to be hauled by tractors a long distance over trails; and delivery could not have been made until too late for the lips to have been of any use that season, as the first steamer to Bethel left Seattle on May 21. Accordingly, the bucket lips were sent by ship to Seward and over the Alaskan Railroad from Seward to Anchorage. From there the Alaskan Airways carried them into the interior and were able to leave them within a short distance of the dredge. All of the bucket lips were delivered in plenty of time for the dredge to commence digging as soon as the weather permitted. Had there been no airplane service, the loss to this operator would have been serious, amounting to practically an entire season's work.

Another instance was when a shipment of dynamite and fuse, destined for Medfra on the Kuskokwim River, missed the last steamer from Seattle. This material would have gone from Seattle to Bethel and transhipped there to a river boat for Medfra. With the close of water transportation there would have been nothing to do but wait for favorable conditions and haul the material in with dog teams. However, as airplane service was available this shipment reached its destination in ample time for Anchorage, and there was no loss of time at the mine. Use of dog teams to serve this section of the country appears to be almost a thing of the past now that aerial transportation is available.

Gold bullion is regularly handled by airplane from many interior points to Anchorage and Fairbanks for shipment to the outside. This means much to the operators because there is no loss of time in getting the gold out.

Freight Transportation in Canada

Airplanes continue to play an important part in the development of new mining districts in Canada. Although air transportation of heavy freight has not been undertaken to the extent that it is employed in New Guinea, the use of airplanes capable of carrying from 1,200 to 2,000 pounds of freight has been perhaps more extensive in the mining districts of Canada than in any other area of the world.

These airplanes have been employed mostly in the earlier stages of prospecting and development to transport men, diamond-drilling outfits, food, and other supplies into the newer districts. As a rule, the planes have been equipped with floats during the summer months and with skis for winter operations. Much of the flying has been done by chartered equipment belonging to air-transport companies, although many routes in mining districts have been flown on regular schedules by air-transport operators carrying passengers, mail, and express.

During the season of 1932, Canadian Airways, Ltd., added to its equipment a single-engine Junkers Ju-52 plane capable according to the manufacturer, of carrying a pay-load of 4,950 pounds for a distance of slightly more than 600 miles without refueling when equipped as a land plane. Float equipment doubtless would reduce the pay-load or the cruising radius, or both. Such equipment should make possible the transportation of heavier loads at lower rates, and its purchase indicates anticipation of additional demand for the transportation of heavy freight into mining districts.

Great Bear Lake District. - With regard to aerial transportation in the Great Bear Lake area, the Canadian Mining Journal for July, 1932, contained these items:

From March to September, 1931, Spence-McDonough's three airplanes covered a distance of 53,500 air-miles in the mining districts of northern Canada. They carried 40,000 pounds of pay freight between Waterways, northern Alberta, and Great Bear Lake. Included in this work was the transport of a complete diamond-drilling outfit to the most northerly point at which such work had been done on the continent. None of their ships so much as damaged a brace during the season, although much of the flying was done under extremely unfavorable conditions.

From Waterways to Great Bear Lake is approximately 300 miles. The 20 tons of freight mentioned was transported by air at a maximum of 80 cents per pound, or \$2 per ton-mile, including complete insurance coverage on the freight.

Water transportation from Waterways to Great Bear Lake is open for only 2½ months during the year. The boat rate has been approximately \$320 per ton, or 40 cents per ton-mile, but 4 to 6 weeks are required for water transportation, as compared with only 12 hours by air.

Ontario. - C. F. Jackson, mining engineer with the United States Bureau of Mines, contributed the following note regarding air transportation for the Ashley gold mine, Matachewan district, Ontario:

Although most of the heavy material and equipment for the Ashley gold mine was hauled in by winter road, 45 tons were transported by airplane during 1932. This freight was flown from Elk Lake (a widening of the Montreal River) to Ashley Lake (formerly called Whitefish Lake), three miles from the mine. The distance between landings is approximately 32 miles. A 4-place, 320-hp. Bellanca with a cruising speed of approximately 100 miles an hour and a pay-load with floats of about 1,150 pounds was used for the freight transportation. The trip required 20 to 25 minutes.

The plane was operated by General Airways, Ltd., with a passenger fare of \$10 one way (31 $\frac{1}{4}$ cents per passenger-mile) and a freight charge of 6 cents per pound, (\$3.75 per short ton-mile). Surface transportation cost for 1,412 tons hauled over winter roads was 64.6 cents per short ton-mile, or about 17 per cent of the air-transportation cost.

Freight Transportation in Mexico

G. C. Quick, pilot for El Paso Air Service, contributed the following, under date of June 16, 1932:

I am operating for the El Paso Air Service a Bellanca Pacemaker equipped with a Wright J-6 motor, over an 80-mile run from Mazatlan to Contra Estaca, Sinaloa, Mexico. I find that it is not advisable to land on my field at the Mexican Candelaria Mining Company at Contra Estaca after 10 a.m., as it is a mountainous country and my approach is poor; the field being an improvised one, 1,750 feet long and 200 feet wide, made by a bend of the Elota River, and situated between steep canyon walls.

My operating cost per flying hour averaged \$27.50 U. S. Currency on this particular run, making an average of 40 trips monthly. The flying time each way averaged 38 minutes, making the average flying time per trip 76 minutes.

Miscellaneous freight including drill steel and structural steel pieces weighing as much as 1,500 pounds, also fresh vegetables, groceries, and supplies, are flown to the mine, and I carry gold and silver bullion back to Mazatlan; also such other miscellaneous cargo as it may be necessary to ship. Passengers are also carried. I hauled approximately 100 tons from January 1 to June 16, 1932. Operations have been perfect.

The only other existing means of transportation to this particular mine is by pack train, which takes 5 days each way as compared with 38 minutes flying time as stated above. I know no other make of plane that would be better suited for this particular run.

The flying cost on this run is given as 29.03 cents per ton-mile. Apparently, this figure does not include depreciation, insurance, or pilot's salary, although full details could not be procured.

Durango. - During the 7 months prior to November 1, 1932, the San Luis Mining Co. of Tayoltita, Durango, transported 198.4 short tons of freight by airplane. This included supplies to the mine and bullion from the mine to the outside. The air-route is approximately 84 miles long, making 168 miles flown on a round trip.

Freight Transportation in the United States

Airplanes have been used for the transportation of freight into mining and petroleum districts of the United States to a much less extent than in Canada and Alaska. In certain isolated districts where ground transportation facilities are poor and the weather in general is favorable, the possibilities appear to be excellent for further development of economical air transportation of heavy freight for mining and petroleum operations.

Aircraft have been used by petroleum and mining companies of the United States for transportation of executives and salesmen, for emergency transportation of repair parts and supplies, and for advertising. However, as yet they have not been employed to any great extent for the transportation of heavier freight.

It should be pointed out that most air-transportation developments in the United States have been directed to a steady increase in speed, with important sacrifices being made to attain this objective. A brief study of air-transportation problems makes one reluctant to challenge the wisdom of these sacrifices for commercial air-line operations. It should be indicated, however, that in the air transportation of freight for mining or petroleum operations, the cruising speed of the aircraft is of minor importance when compared to such factors as weight-carrying capacity, stability, dependability, slow-landing characteristics and, in some instances, ability to land and take-off from small, rough landing fields, and to climb rapidly to avoid obstructions in the take-off.

A contribution from Newton Wakefield, mining engineer, vice-president and general manager, Maner Air Transport, on available flying equipment is summarized as follows:

There are many types of airplanes on the market, ranging from the fast single-engine to the large single and multimotored transport planes. There are land planes, seaplanes, and amphibians, all of which have their special uses. There are comparatively few, however, that are suited for transportation of heavy machinery for mining and petroleum operations. The choice of equipment depends in each instance on the work to be done which imposes certain limitations upon the types of planes to be used. The restricted area of landing fields eliminates the use of planes with fast landing speed. The competition of surface transportation usually makes it impractical to use high-speed equipment where pay-load is sacrificed for speed. The comparatively high altitude of some landing fields imposes further restrictions.

Rather than to go into a technical discussion, it may be stated in general that planes used in mining and petroleum transportation work should have the following characteristics, which will be varied somewhat with the weight of single loads to be transported and the local conditions of operation:

Comparatively slow landing-speed, high rate of climb, large pay-load, economy in operation, comparatively long cruising range, sturdy construction and ease of loading and unloading are important factors to be considered. In general, all-metal construction has received the preference over metal and fabric construction, particularly where heavy loads are to be carried and where hangars are not available. While fairly high cruising speed is desirable under certain conditions, it is by no means essential, and other factors are far more important.

Idaho. - It appears that more mining supplies and freight have been transported by air in the Idaho-Montana Districts than elsewhere in the country.

The Idaho Sunday Statesman for May 8, 1932, contained an article entitled "Winged Freighters" by Bert Eustis, from which the following information is summarized:

During the winter of 1931-32, airplanes regularly supplied food and news of the outside world to a number of isolated and snow-bound mountain villages in Idaho.

The service was maintained by ski-equipped planes operated by pioneer pilots without the benefit of beacons and lighted, clearly marked, well-tended airports. There were no beam radios to assist the pilots in keeping on their routes which lay over rough mountain peaks. The doubtful landing fields are thousands of feet above sea level where the take-offs are difficult and sometimes almost impossible. Snow clouds hang low almost all winter and the air is rough and full of pockets. Only a few minor mishaps occurred, such as damaging propellers or skis on landing. Passengers and mail have been taken in and out of the villages. Express and food have been flown in to relieve distress. Necessary freight has been hauled to the mines to permit them to continue operation and supply employment. Injured miners have been rushed from isolated districts to the more efficient hospitals of the larger cities.

United States Forest Service: The freight transportation problems of some mining and petroleum operations in isolated districts are not dissimilar to those of the United States Forest Service. Hence, the following summary, prepared from information furnished by the northern district of the United States Forest Service may be of interest:

During the 1931 season 129,413 pounds of freight was delivered by airplane an average air-line distance of 60 miles from Missoula, Montana, at a contract price of 5.5 cents per pound, corresponding to a ton-mile cost of \$1.833. The materials ranged from eggs to ready-cut buildings. All freight was delivered in excellent condition without mishap, to emergency landing fields in inaccessible mountain areas over passes with an elevation of 7,000 feet or more.

During the 1930 season, 116,000 pounds of similar freight was transported by pack horse at a cost of 7.3 cents per pound, corresponding to \$2.43 per ton-mile or 32.7 percent more than the cost of air transportation.

AIR TRANSPORTATION OF PERSONS IN THE UNITED STATES

The increased use of light aircraft for transportation of officials and supervisory and service employees in the mining and petroleum industries is a highly promising field. During the past two years the development of suitable types of craft for this purpose has been considerably more rapid than may be generally realized. The author is convinced that in many instances the effectiveness of more highly paid employees could be doubled by using suitable aircraft at only a slight increase in the present cost of their transportation. In some instances the present cost should be decreased. Before discussing this phase in detail, however, it is desired to present summaries of individual air transportation costs actually developed for some allied phases of the problem.

Conventional types of larger airplanes with fair cruising speeds have proved reasonably satisfactory for the transportation of company directors and highly paid executives. The indirect cost of such flying has been high, due principally to the small number of hours flown and to the use of large, multi-motored planes. In some instances, flying costs have been increased to prohibitive amounts, (1) by using large, expensive planes only 10 to 15 hours per month, (2) by the continuous employment of pilots, co-pilots, and traveling mechanics for little-used equipment, and (3) by the construction of private flying fields and hangars which are used to only a slight extent.

Flying Costs: The factors that determine the actual cost of air transportation of persons are perhaps more numerous than those which control the cost of automobile transportation. The questions of weight, design and size of equipment, horsepower of engines, first cost and hence depreciation of equipment, hours of use per year, whether stand-by pilots and mechanics are continuously employed, elevation and character of terrain over which operated, weather conditions of area in which operated, the cost of landing-field and hangar facilities, local costs of gasoline, oil, mechanical repairs and check-ups, as well as the character of the equipment and the kind of service in which operated, all naturally have a vital bearing on the cost of flying.

A major petroleum company reported that it operated a Hornet-powered, Boeing 40B mail plane at a cost of \$45 per flying hour or 39 cents per plane-mile, including pilots' and mechanics' salaries and expenses, hangar expense, fuel and lubrication cost, repairs and renewals, taxes, insurance, and depreciation. As this particular ship carries 3 passengers in addition to the pilot, the cost is equivalent to 13 cents per passenger-mile, exclusive of pilot. Insurance is rather expensive as it includes public liability and property damage, in addition to life insurance policies of \$25,000 each on the pilot and three passengers. During a period of 43 months prior to June 1, 1932, the plane had been flown a total of 1,886.4 hours, over a distance of 182,075 miles, and 4190 passengers had been carried.

In Western Flying for January 1933, page 22, the cost of flying a Stinson Junior plane some 65,000 miles during 1932 is given as 15.94 cents per plane-mile, including continuously employed pilot's wages, depreciation, insurance, fuel, maintenance, and contingent items. The plane was used in north-central Nebraska in connection with ranch operations.

W. A. Mara, vice president of the Stinson Aircraft Corporation, stated that during 1931, the average operating cost of all the company's district salesmen's 4-place cabin Stinson Jr.-S planes throughout the United States was 8.46 cents per mile. The figures ranged from 6 to 11 cents per plane-mile, depending upon the locality and the amount and character of operation of the individual planes. These figures include servicing and repairs, storage and the purchase of gasoline and other necessary items at any airport where the salesmen happened to have their planes for demonstration.

The Martin-Decker Corporation, an oil-well supply firm with headquarters at Long Beach, California, has furnished estimates of the cost of flying its 4-place Stinson cabin monoplane, with the exception of insurance and miscellaneous storage costs. The Stinson, Junior was purchased in May, 1929, for \$5,775. To October 30, 1932, it had been flown 870 hours at approximately 100 miles per hour at a cost of 7,958 cents per plane mile.

C. F. Williams, district mining supervisor for the United States Geological Survey, headquartered at Miami, Oklahoma, provided a good example of the results attendant upon the use of aircraft in connection with his work. He has used three small cabin monoplanes, fitted with engines ranging from 90 to 110 hp. At present, he is using a Star-Cavalier 90, a 2-place cabin monoplane with a 90 hp. Lambert engine. Between July 1 and December 31, 1931, the plane was flown 18,400 miles, at an operating cost of 3.59 cents per plane mile exclusive of storage, resulting in a time saving of about 30 percent under that required for automobile travel in the same district.

Gasoline and oil consumption.— Some approximate estimates of gasoline and oil consumption at cruising speeds for radial air-cooled engines in common use, as established by practical experience are as follows:

Gasoline and oil consumption of certain engines

<u>Rated hp.</u>	<u>Gasoline, gal. per hour</u>	<u>Oil, gal. per hour</u>
400-420	18 - 25.0	0.66
300	16 - 21.3	.61
240	13 - 14.5	.46
215-220	11.5 - 13.0	0.39 - 0.41
160	8.0 - 10.5	---
125	6.3 - 8.9	.11 - .26
110	7.4 - 8.2	.19 - .45
100	6.0 - 7.0	---
90	5.7 - 7.0	---

Because of the number of variables introduced under actual flight conditions, consumption easily may exceed the higher figures given for each power rating. Fuel consumption in pounds per horse-power-hour is published by the engine manufacturers.

The cost of flying decreases rapidly as the number of hours flown per year increases. For the employee who could make effective use of 400 to 600 hours of air transportation per year the cost per mile should be considerably under that of automobile transportation, and the time saving and increase in effectiveness would be proportionately greater.

It is the opinion of the author that oil companies, in particular, could gradually reduce their costs of field supervision to a considerably extent by establishing their own flight instruction departments and training selected individuals as private pilots. As their courses of instruction are completed, these men could be equipped with suitable light aircraft to be used for individual transportation in connection with their duties in the oil fields, where conditions are favorable. As the hazards and costs of automobile driving are increased considerably when one car is driven by several individuals, so the hazards and costs of flying would be much increased by passing an air-plane from one individual to another for use in the oil fields. Thus, the author is forced to the conclusion that the wiser plan would be to utilize low cost equipment and provide selected individuals with small planes for their sole use.

The adoption of such a plan naturally would incur the building up of mechanical and servicing units within the organization quite similar to the automobile repair and servicing units now functioning.

Although the light airplane has some weather restrictions, probably being held on the ground during bad weather when a car could get through, on the other hand, on fair days following bad weather such as snow or heavy rains and floods that have blocked the roads, the airplane could go through in safety, when automobiles could not get through.

LANDING FIELDS

Many mining operations are conducted in mountainous localities. Not infrequently, however, it is possible without undue expense to get two runways of about 2,000 to 3,000 feet in length and 300 to 400 feet in width at approximately 90 degrees. Sometimes the general direction has to be changed somewhat to suit the prevailing air current. The two runways need not be connected and can be a mile or two apart if necessary, if they are both kept in fairly close proximity to the loading point. If, as frequently happens, the operation is in rough country, the construction of a landing field may present a serious problem.

If the mining or petroleum operations are on or near a body of water, seaplanes and float-equipped land planes can be used successfully for transport work. In general, however, the pay-loads of planes so equipped are less than those of landplanes of similar gross weight. Suitable locations, of course, are not numerous in mountainous districts. The number of available places for

the construction of landing fields at comparatively low cost in many areas, however, is rather surprising. Places that justify careful examination can be found most easily as a rule by flying over the area. Flat benches, sand-bars, and meadows frequently make good fields. Graded waste and tailings dumps have been used to advantage.

The United States Forest Service has constructed some landing fields in inaccessible localities in the national forests of the Northwest. They include some good fields in the Selway, Idaho, and Flathead National Forests. They are available, of course, for mining companies in the districts as concentration points to receive supplies and materials for development work in the vicinity.

The Forest Service has prepared specifications for air transport landing fields to provide for the transportation of forest fire-fighters. These specifications might be of service to mining or petroleum companies contemplating the air transportation of freight.

It has been stated that on occasion the Forest Service has found that the cost of clearing and preparing a landing field suitable for transport airplanes in forest areas has been comparable to the cost of building a mile of trail through those areas. In a special instruction circular issued by the Forest Service, the following vital features are recommended for consideration in selecting fields for transport planes.

1. Since changes in direction of flight are to be avoided in the first half mile from the field, locations should be chosen so as to exclude major obstacles so far as possible within a half-mile radius of the outside of the field.
2. Obstacles in the direction of the prevailing wind are especially bad.
3. The cleared ground should be nearly level. Slopes of $2\frac{1}{2}$ percent are not objectionable. Slopes up to 5 percent are acceptable but not desirable.
4. The runway must be reasonably firm and so surfaced that a car can be driven over it at 25 miles an hour. A firm, smooth surface is highly important for a heavy plane.
5. The approaches to the field must be up to the specifications shown in the attached sketch (sketch omitted).
6. A runway at right angles to the main runway is desirable, it may be in a second field as near to the first as the conformation of the ground permits. The availability of this secondary runway at right angles to the first is of increasing importance where the direction of the wind varies over wide limits. Conversely, where the prevailing direction of the wind is fairly constant, the importance of the secondary runway decreases.

DEPENDABILITY OF AIRPLANES

Weather conditions are of paramount importance in considering the maintenance of flying schedules in comparison with the maintenance of schedules for ground transportation. There is considerable variation in percentage of schedules actually flown as against those planned for different sections of the United States. Some idea as to allowances to be made for delays due to climatic conditions can be obtained by studying Aeronautics Branch figures for miles scheduled and miles actually flown for various sections of the United States. Such figures are published from time to time by the Aeronautics Branch of the Department of Commerce in its AIR COMMERCE BULLETIN.

Although means of ground transportation may be continued during weather unsafe for flying operations, it should be noted that immediately upon the break-up of a storm, flying may be resumed if landing fields are in usable condition. Roads and trails, on the other hand, may be blocked by snow, landslides, fallen timber, and other results of storms so that a much longer time would be required to resume ground transportation. Thus, where climatic conditions are unfavorable, actual delays in truck, tractor, pack-train, or dog-team transportation may be considerably greater than the delays entailed in grounding flying equipment during the height of the storms.

A brief review of miles flown per accident for scheduled air transports as well as for miscellaneous flying operations in the United States, according to the Air Commerce Bulletin, will give a rough idea of the dependence which the mine or petroleum operator may place in his specially operated air-transport facilities.

Figures released by the Assistant Secretary of Commerce for Aeronautics in Air Commerce Bulletin, May 1, 1933, show that in scheduled air transport operations 5,862,103 passenger-miles were flown per passenger fatality during 1932. In miscellaneous flying operations the miles flown per passenger fatality were 587,810 during 1932, as against 623,514 for 1931. The miscellaneous flying operations designation included student instruction, experimental flying, miscellaneous commercial flying, sight-seeing, dusting crops, aerial photography, and pleasure flying.

On the basis of figures announced by the Aeronautics Branch, it appears that the average air-line passenger could travel around the earth 1,000 times on scheduled air-lines before he would run a chance of becoming involved in a fatal accident, or he could travel on an air-line for 481 years before he would be involved in a fatal accident.

GENERAL SUMMARY

1. Airplanes can be made to reduce the time and cost of prospecting and exploration work, particularly in remote areas. They can increase the safety and effectiveness of men employed on exploration, prospecting and mineral development operations. The employment of mining and petroleum engineers and geologists who have been thoroughly trained as pilots is recommended.

2. The choice of suitable flying equipment for mining and petroleum operations is important. High cruising speed is relatively unimportant for most airplanes used in these industries. Such characteristics as dependability, stability, comparatively low landing speed, quick take-off, high rate of climb, large pay-load, economy of maintenance and operation, comparatively long cruising radius, sturdy construction, and similar characteristics which would be varied to fit conditions are far more important than high speed.

3. Aerial surveys can be made to expedite geological and engineering work in connection with the development of mineral and oil deposits. When properly made they increase the accuracy and reduce the cost of geological and engineering as well as development work. To be effective they should be made by men thoroughly trained and experienced in aerial methods, particularly in the accurate interpretation of aerial photographs.

4. Suitable aircraft capable of cruising safely at low speeds would be especially valuable for geological and engineering reconnaissance and similar work including quick and accurate location and routine inspection of pipelines, long aerial tramways, power transmission and telephone lines, location of trails, roads, highways, railroads, camps, available water power, water supply, timber for mining and construction purposes, and similar auxiliary projects.

5. When the estimated cost of ground transportation of freight exceeds 11 cents per ton-mile, when the ground transportation cost of individuals exceeds 5 cents per mile, or when available ground methods are slow or otherwise unsatisfactory, consideration should be given to the use of aerial transportation.

6. Even when ground transportation is fairly cheap and satisfactory, auxiliary air transportation should be considered for executives, sick or injured persons, bullion, high-grade concentrates, perishables, emergency repair parts, emergency supplies, samples, drill cores, and similar items. Airplane ambulance service is of particular importance in remote areas.

7. In many instances the cost of mining and oil operations could be reduced and effectiveness could be increased to a considerable extent by increasing the use of suitable airplanes, under favorable conditions.

8. The training of selected employees as private pilots and the substitution of suitable airplanes for various forms of ground transportation under favorable conditions would reduce supervisory, operating, transportation, and other costs while increasing effectiveness in the industries under consideration. In many instances the total cost of operating suitable light pilot-yourself aircraft should be less than the present cost of operating automobiles.

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MANGANESE; ITS OCCURRENCE, MILLING, AND METALLURGY

PART I.

CHAPTER 1. PHYSICAL PROPERTIES AND PREPARATION OF
METALLIC MANGANESE

CHAPTER 2. MANGANESE IN NONFERROUS ALLOYS

CHAPTER 3. COMPOUNDS OF MANGANESE

CHAPTER 4. OCCURRENCE AND ORE-DRESSING POSSIBILITIES
OF ORES OF MANGANESE IN THE UNITED STATES



BY

R. S. DEAN, FRED D. DEVANEY AND WILL H. COGHILL

INFORMATION CIRCULAR
UNITED STATES BUREAU OF MINES

MANGANESE; ITS OCCURRENCE, MILLING,
AND METALLURGY¹

Part I

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INTRODUCTION

This circular presents part I of a summary of the occurrence, milling, and metallurgy of manganese in the United States. An attempt has been made to assemble the data on manganese and its metallurgically important compounds which have been obtained in several years of research in the Bureau of Mines laboratories, as well as most of the published data of other investigators. Parts II, III, and IV will contain chapters on thermodynamic properties and metallurgy. Part V comprises the bibliography and index.²

The experimental work reported herein was done in cooperation with the University of Missouri School of Mines, Rolla, Mo.; the University of Minnesota, Minneapolis, Minn.; the Carnegie Institute of Technology and the Metallurgical Advisory Board, Pittsburgh, Pa.; and the University of Nevada, Reno, Nev.

² Throughout parts I to IV, numbers in parentheses refer to items in this bibliography, and page numbers to those in the reference cited.

CHAPTER 1. PHYSICAL PROPERTIES AND PREPARATION OF METALLIC
MANGANESE

By R. S. Dean³

PURE MANGANESE

Although manganese is one of the commoner metals, as judged by the distribution of its compounds in the earth's crust, comparatively little is known about the element itself, due to the difficulty of obtaining the metal in a high state of purity. Manganese of more than 99 percent purity has been available only since the introduction of vacuum distillation in the high-frequency furnace by Gayler (107) in England and Friauf (98, 255) in this country.

Table 1, from Gayler (107), gives the analyses of distilled manganese.

TABLE 1. - Analyses of Distilled Manganese, Percent

Sample	Sulphur	Silicon	Carbon	Iron	Alu- minum	Phos- phorus	Mag- nesium	Man- gane
A	-	Trace	0.013	Nil	Nil	-	-	-
B	-	Nil	.01	do.	do	-	-	-
C	-	0.021	-	do	Tr.	-	-	-
D	-	.005	-	do	do	-	-	-
E	-	.11	-	do	do	-	-	-
F	0.012	.06	.01	Under 0.04		0.01	Nil	1/99.5

1/ By difference, 99.87.

NOTE.- On similar samples the nitrogen content was found to be 0.01 percent or less.

Manganese of this purity as distilled or after vacuum melting is reported by both Gayler and Friauf to be silver-gray and very brittle. It is hard enough to scratch glass and will not tarnish in air.

The question whether the absolutely pure metal is ductile must remain unanswered. Corson (59) believes that the pure metal is ductile. He bases this opinion on the fact that Zhemchuzhny was able to obtain ductile manganese and draw it to fine wires by the simple expedient of adding about

³ Chief engineer, Metallurgical Division, U. S. Bureau of Mines.

3 percent copper to the aluminothermic product. Corson suggests that the copper brings the silicides, aluminides, nitrides, and carbides into a less harmful form of distribution.

Dyson (78) describes manganese prepared by electrolysis of aqueous chloride solution as having a faint pink tinge and being soft and easily polished. The annual report of the National Physical Laboratory for 1924 states that the product of such an electrolysis is subject to disintegration and oxidation, indicating that it is essentially different from the distilled product. Campbell (44) found electrolytic manganese brittle even when annealed to remove hydrogen.

These differences with regard to the properties of metallic manganese find an explanation in the existence of several allotropic forms of the element. Gayler (107) has found four transformations accompanied by marked thermal phenomena occurring at the following temperatures.

	$^{\circ}\text{C.}$		$^{\circ}\text{C.}$
(1)	$1,191 \pm 3$	(3)	742 ± 1
(2)	$1,024 \pm 2$	(4)	682 ± 1

Microscopic examination indicates that transformation 3 is accompanied by a change in crystal structure, while no such change is apparent with regard to transformation 4. Moreover, one of these transformations is accompanied by a marked change of volume, which is so great that an ingot formed by melting the distilled metal in vacuo is cracked extensively on cooling.

Westgren and Phragmen (388) have shown by X-ray spectrometry that electrolytic manganese is tetragonal with

$$a = 3.774 \pm 0.003\text{\AA}; c = 3.533 \pm 0.003\text{\AA}; c/a = 0.937.$$

Since the manganese produced by distillation has a cubic lattice, this explains sufficiently the difference of properties between distilled and electrolytically deposited manganese without recourse to assumed effects of impurities. The investigations of Perrson and Ohman (303) show that by alloying manganese with small amounts of copper and quenching from temperatures above $1,160^{\circ}\text{C.}$ structures showing the tetragonal lattice of electrolytic manganese may be obtained. By varying the amount of copper and extrapolating to pure manganese the parameters of this γ manganese are obtained as

$$a = 3.776 \pm 0.005; b = 3.525 \pm 0.008; c/a = 0.934.$$

By comparing these with the parameters given by Westgren and Phragmen quoted above, it is evident that this high-temperature form of manganese is identical with the electrolytic form. Perrson and Ohman also found that this high-temperature form, when stabilized by small amounts of copper, is soft and ductile, which is in agreement with the observations of Zhemchuzhny.

The γ form of manganese, whether formed by electrolysis or quenching is rapidly transformed to α manganese when heated to 150°C. and even at room

temperature is completely changed after a few weeks. This instability is doubtless responsible for the variation in properties described.

The transformation found thermally by Gayler (107) at $1,024^{\circ}\text{C}$. was not found by Perrson and Ohman (303) in their study of crystal form as revealed by X-ray methods. Neither was it found in the further work of Gayler (108).

According to Schulze (343), commercial manganese is frequently found to be a mixture of the α form with another very complicated structure. By heating this commercial manganese to $1,000^{\circ}\text{C}$. and quenching, X-ray studies show only the second complicated structure which has been called β manganese. This structure is a cubic lattice with 20 atoms to the unit cube, while α manganese has 56 atoms to the unit cube. The transformation $\alpha \rightleftharpoons \beta$ is identified with the thermal change found by Gayler at 742°C . The transition at 682°C . is unaccompanied by a change in structure. Bradley (33) suggested that the two forms of manganese occur in a state of isodynamic isomerism within the temperature interval $650^{\circ}\text{--}850^{\circ}\text{C}$. and that the equilibrium gradually shifts over to the β modification as the temperature is raised. Perrson and Ohman (303), however, found no trace of β manganese in the pure metal quenched from 730°C ., while that quenched from 760°C . showed only faint α lines.

Bernal (23) has proposed that both α and β manganese are alloys and points out that their structures are such that if manganese were not known to be an element they would certainly be counted as alloy structures. β manganese is, in fact, isomorphous with AlAg_3 . The sudden break in the atomic volume curve at manganese is adduced as further evidence that both α and β manganese may be, in fact, alloys. Additional support for this view is found in the peculiar behavior of manganese during work hardening by the pendulum tester. Sandifer (334) reports that the hardness falls from 77 to 30 as the pendulum swings some 40 times. This effect may be due, however, to cracking.

In summary then, manganese exists in three well-defined crystalline varieties⁴:

α	= 56 atoms per unit cube;	a	= 8.89; stable below 742°C .
β	= 20 atoms per unit cube;	a	= 6.28; stable $742^{\circ} - 1,191^{\circ}\text{C}$.
γ	= tetragonal	a	= $3.774 \pm 0.003\text{\AA}$; stable $1,191^{\circ}\text{C-M.P.}$
		c	= $3.533 \pm 0.003\text{\AA}$.
		c/a	= 0.937.

The α and β forms are brittle at room temperatures. The β form becomes slightly malleable at 900°C . The γ form is soft and ductile. By electrolysis the γ form is obtained in a metastable state.

The melting point of manganese is given by Gayler as $1,244 \pm 3^{\circ}\text{C}$., which is some 20° higher than that obtained by earlier investigators and is presumably to be accepted as accurate, because of the high purity of Gayler's sample, although this value stands otherwise unsupported. The vapor pressure,

⁴ Values for the parameters taken from Ewald and Hermann's Struckturbericht.

specific heat, and other thermodynamic properties of manganese are discussed in part II, chapter 5.

The specific gravity is variously given, depending on the sample used. Figures ranging from 7.05 to 8.01 have been recorded. The true density of α manganese from X-ray measurements is given by Preston (309) as 7.44. The atomic volume of manganese is given by Goldschmidt (118) as $14.85 \cdot 10^{-24}$.

The volume change on solidification of manganese has been investigated by Matsuyama (245), who found that a contraction of 1.7 percent, a comparatively small shrinkage, occurs on solidification. The figure for copper is 4.2 aluminum 6.26, silver 5.00, and iron 5.5. For detailed consideration of manganese in various states see chapter 5.

The initial linear compressibility of manganese is given by Bridgeman (40) as $2.6 \cdot 10^{-7}$. This figure is about normal with respect to the position of manganese in the periodic table.

The electrode potential of presumably α manganese analyzing 99.6 percent manganese, prepared by electrolysis of manganese salts into a mercury cathode and subsequent distillation of the mercury and melting in hydrogen, is given by Royce and Kahlenberg (321) as 1.45 volts, based on the value +0.560 for the normal calomel electrode. The magnitude of this potential is confirmed by the replacement of zinc, cadmium, iron, cobalt, nickel, lead, hydrogen, copper, and silver from their solution by metallic manganese.

The atomic electrical conductance of manganese is given by Kraus (214) as 0.6561. This and most other figures in the literature for the electrical conductance of manganese seem to be based on the work of Kuh (222). Stated in terms of specific resistance, Kuh's value is $82 \cdot 10^{-6}$ ohms per cubic centimeter at room temperature. Hume-Rothery (174), however, gives $5 \cdot 10^{-6}$. No data are available for the temperature coefficient.

The magnetic susceptibility of manganese is given by the International Critical Tables as $9.9 \cdot 10^{-6}$ at 18°C . More recent figures obtained by Mary H. Wheeler (according to a letter from Prof. L. W. McKeehan, Yale University), are $10.58 \cdot 10^{-6}$ for α manganese and $9.70 \cdot 10^{-6}$ for β manganese. Accordingly, it is one of the more strongly paramagnetic metals. Kuh states that manganese fused in hydrogen is ferromagnetic. At $H = 12,000$ Oe; $H_c = 670$; and $d = 7.24$. (The B/H curve for this manganese is given in the International Critical Tables, vol. 6, p. 409, fig. 47.)

Freeze has confirmed Kuh's findings concerning the ferromagnetism of manganese fused in hydrogen, but found that electrolytic manganese or films of manganese prepared by sputtering in hydrogen were not ferromagnetic.

Oschenfeld (286) finds that manganese is made ferromagnetic by heating above $1,000^\circ\text{C}$. in nitrogen, whereby a solid solution of nitrogen in δ manganese is formed which is stable at room temperature.

Preparation of Metallic Manganese

Manganese is prepared commercially by heating manganese oxide with aluminum. Gayler (107) reports that if Mn_3O_4 is used it can be more easily ignited and better results obtained. Hadfield (140) gives the analyses of commercial metallic manganese shows in table 2. This method is due to Goldschmidt and Vautin (117).

TABLE 2. - Analyses of Commercial Metallic Manganese, Percent

Manganese ..	90.35	90.50	92.20	95.50	96.00	1/ 97.85
Carbon	1.18	.78	1.04	.08	.06	.025
Silicon92	.49	4.96	1.40	.60	.64
Sulphur	-	-	-	-	.208	-
Phosphorus .	.27	.30	0.36	-	.068	-
Aluminum ...	-	-	-	-	1.03	.28
Iron	-	-	-	-	.78	.75

1/ This was a special high-grade manganese from Thermit, Ltd.

Debuigne, in British Patent 221233, August 31, 1923, claims that the addition of oxides or hydroxides such as $Ca(OH)_2$ to the aluminothermic mixture lowers the reaction temperature and prevents reduction of SiO_2 .

The electrodeposition of manganese may be accomplished from either sulphate or chloride solutions in water provided sufficient ammonia is present to neutralize the acid formed (91).

According to Allmand and Campbell (5) the best conditions for the electrodeposition of pure manganese are the electrolysis of a solution containing manganous and ammonium sulphates separated by a diaphragm from the anolyte (ammonium sulphate solution) the hydrogen ion concentration being kept at 10^{-6} - 10^{-8} by regulated additions of sulphuric acid or ammonia. The temperature is $30^\circ C.$, the cathode current density 10-15 amp./dm². The rotary aluminum cathode has a burnisher pressing against it.

Electrolytic manganese is the unstable δ modification, the properties of which have already been described. Its density is considerably lower than for the distilled metal. Campbell gives the density as 7.034.

Electrolysis from fused silicates containing some iron gave only iron with 0.8 percent of manganese (45, 271).

CHAPTER 2. MANGANESE IN NONFERROUS ALLOYS

By R. S. Dean⁵

MANGANESE AS A SCAVENGER

Manganese has been said to be the universal cure-all and tonic for the metallurgist. It appears in the formula for almost every alloy that has been patented or described in detail. The full explanation of this effect of manganese is by no means available; however, the position of manganese as a transition element between the highly electropositive elements and the heavy metals apparently makes it valuable.

Thus manganese is sufficiently electropositive to have a high affinity for sulphur and a sulphide of essentially nonmetallic properties, yet the metal is not too volatile or easily oxidized to be readily used as an addition. Furthermore, manganese forms solid solutions with most metals so that the excess added does not occur in a particularly harmful form. The relative affinity between metals and sulphur is given by Parravano and Malquori (292) in the following list: $Ag < Pb < Ni < Fe < Cu < Mn$.

Moreover, the fact that manganese sulphide is insoluble, whereas heavy metal sulphides are somewhat soluble in metals enables the desulphurizing reaction to proceed to completion. The physical properties of the manganese sulphide formed are also important. According to Picon (304) it vaporizes without decomposition at 1,375°F. so that when it appears as a free phase in nickel and copper alloys, considerable loss by vaporization is to be expected. Manganese sulphide is soluble in silicate slags. The system $MnS - MnSiO_3$ has been investigated by Wolesskow (394) and by Glaser (114). Both investigators report a eutectiferous system. If the alloy is melted under a silicate slag the manganese sulphide may therefore find its way into the slag.

COMMERCIAL ALLOYS CONTAINING CONSIDERABLE PERCENTAGES OF MANGANESE

Only the alloys of manganese with copper and aluminum have great importance commercially.

System Copper-Manganese

Since the axial ratio of tetragonal manganese approaches 1 with increasing copper content manganese can form solid solutions throughout the whole range of composition with copper, which has a face-centered cubic structure. In general, the mechanical properties of the copper-manganese alloys up to

⁵ Chief engineer, Metallurgical Division, U. S. Bureau of Mines.

30 percent manganese change continuously with increasing manganese content. All these alloys, according to Patterson (296) are face-centered cubic with a increasing from 3.60-3.70A (296). According to Corson (59) they may be melted in a graphite crucible without absorbing more than 0.2 percent carbon, which is not objectionable except for heavy drawing. According to the same author, the hardness of such carbon-containing alloys is sharply dependent on the state of subdivision of the manganese carbide. By quenching from 800°C. and reheating to 400 to 450° for 24 hours, a hardness of 125 may be obtained compared to about 40 for the annealed alloy.

Manganese raises the annealing point of copper, hence alloys containing 4 to 5 percent manganese are used for turbine blades; 4 percent manganese is also used in copper for locomotive fire-box plates and stays, where its principal function seems to be to insure complete deoxidation, with consequent freedom from brittleness caused by reducing gases.

The greatest interest in the copper-manganese system, however, arises from the fact that the alloys containing 12 to 17 percent manganese have a very low temperature coefficient of electrical conductivity. Such alloys, known as manganin (8 to 12 percent manganese), have been described by Bash (16).

The low melting point (about 850°C.) of the alloys containing 20 to 35 percent manganese and the strong but chemically neutral skin of oxide covering the molten alloy have led Corson (59) to suggest their use as die-casting alloys. U. S. Patent 1671408 covers alloys containing 15 to 25 percent manganese and 1 to 3 percent phosphorus and silicon.

The high-manganese members of the copper-manganese system are especially interesting, because the addition of a few percent of copper enables the γ form of manganese to be obtained at room temperature by quenching. Since, according to Perrson (301), no solubility of copper in α or β manganese can be detected, the copper must be rejected as a separate phase when the gamma form of manganese breaks down to the α and β .

Ternary Copper Alloys Containing Manganese

These alloys include special brasses, manganese bronze, and Heusler alloys.

In general, manganese acts as a substitute for copper when added to brass, so that alloys containing 30 percent zinc and 10 percent manganese would have properties similar to those of cartridge brass. Such alloys have not, however, found commercial use. The so-called manganese bronzes, containing about 1 percent manganese in addition to iron, silicon, tin, or aluminum, owe their value to the introduction of other elements, notably iron, with the manganese. A brass has recently been developed for hot-pressing, in which silicon is added with the manganese to such an extent that plainly visible crystals of hard manganese silicide are embedded in a body of regular manganese bronze.

The alpha-phase boundary of the ternary system copper-silicon-manganese has been carefully determined by C. S. Smith (351), who finds that the solubility falls to less than 0.5 percent at 450°C., when more than 4 percent manganese or silicon is present.

Another most interesting series of ternary copper-silicon-manganese alloys are the so-called Heusler alloys. These are alloys of copper with 7 to 11 percent aluminum and 20 to 30 percent manganese. By quenching these alloys from a high temperature and drawing at 600o to 700°C. followed by quenching they acquire ferromagnetic properties.

The structure of these Heusler alloys is given by Elam. (81) as follows:

Mn	Al	Cu	
27.7	5.7	66.6	Face-centered, $a = 3.7$.
28.2	8.6	63.2	Body-centered, $a = 2.975$.
27.1	12.3	60.2	Body-centered, which is cubic, 58 atoms per unit cell.

No commercial uses have been found for these alloys.

Silver can be substituted for copper in Heusler alloys, according to Potter (308), the silver, aluminum and manganese alloys being ferromagnetic. The best alloy is 5 atoms of silver, 1 of manganese, and 1 of aluminum; it has a saturation in tensivity of 70 units per cubic centimeter or somewhat less than Heusler alloys.

The special brasses containing nickel and manganese are said to have low corrodibility. A typical composition as given by LeThomas (232) is Cu, 53.2 percent; Zn, 38.8 percent; Ni, 4.46 percent; Mn, 3.0 percent; Fe, 56 percent (31).

Commercial manganese - aluminum alloys contain 0.5 to 3.0 percent manganese. Manganese is also used as a constituent of duralumin-type alloys where, according to Corson (59), it always forms the compound Al_3Mn . This, however, has not been confirmed by Dix (72). According to Meissner, manganese is not indispensable but up to 0.6 percent manganese considerably increases strength.

The alloy of aluminum with 2.5 percent manganese, 2.25 percent magnesium, and 0.2 percent antimony has been found to be particularly effective in withstanding corrosion of sea water. It is known in Germany as K. S. Seewasser.

According to Haas (132), an alloy of 2 percent manganese is to be recommended for sand castings. It has a tensile strength of 30 kilograms

per square millimeter and an elongation of 5 percent. For rolling, the alloy should contain 1.25 percent manganese, which has a tensile strength of 29-81 kilograms per square millimeter, and an elongation of 25 to 1 percent, depending on mechanical and heat treatment.

The structural details of the aluminum-manganese system have been investigated by Bradley and Jones. (34). They found that a small amount of aluminum added to manganese lowers the β and γ transition point of manganese to such an extent that the γ form is stable at room temperature. Seven different phases are found to exist.

Manganese Alloys of Minor Commercial Importance

The magnesium-manganese alloys have been investigated by Schmidt (338), who concluded that the manganese appeared as an intermetallic compound. Bakken and Wood (13) report that manganese is soluble up to at least 3.2 percent. Pearson (298) concludes from microscopic investigations that the solid solubility extends beyond 2.7 percent.

Nothing seems to be known on the physical properties of these alloys. Manganese is added to magnesium-aluminum alloys as a corrosion inhibitor.

Investigations of the nickel-manganese alloys have been summarized by Pilling and Kehlgran (305), who give them as a nearly continuous series of solid solutions with a minimum melting point at 1,000°C. A heterogeneous range between 41 and 70 percent manganese is suggested, but little is known of its exact nature.

Blumenthal, Kussmann, and Scharnow (27), in a more recent investigation of the alloys of manganese and nickel and of manganese and cobalt have reported that both alloys are heterogeneous. According to Kaya and Kussman (194) a distribution of manganese atoms in the nickel lattice corresponding to the formula Ni_3Mn makes alloys strongly ferromagnetic.

The mechanical properties of the nickel-manganese alloys have been investigated by Mudge and Luff (265). They conclude that manganese up to 4.63 percent results in a definite increase in strength, toughness, and hardness of the nickel alloy, with practically no decrease in ductility. The nickel-copper alloys are similarly affected. Alloys of 96 percent nickel and 4 percent manganese are used for the points of spark plugs.

The manganese-zinc alloys have been investigated by Ackermann (1), who found that zinc alloys with more than 3 percent manganese are brittle. According to Parravano and Montoro (293) at least three solid-solution phases are present in zinc-manganese alloys up to 23.3 percent manganese. Up to 0.98 percent manganese the η phase has a hexagonal lattice like that of zinc. Above this amount the ϵ phase forms, having a hexagonal lattice different from zinc and finally a γ phase with a body-centered cubic lattice forms.

Lead and manganese do not alloy, but the addition of manganese to alloys of lead, antimony, and tin has been covered in British Patent 309399, January 9, 1928. It is suggested as having value as a bearing metal.

The system silver-copper-manganese has been investigated by Keinert (199). Two series of solid solutions are formed. The silver-rich alloys can be hardened at 280°C. after quenching from 750°C.

CHAPTER 3. COMPOUNDS OF MANGANESE

By R. S. Dean⁶

METALLIC COMPOUNDS

G. Haag (130) has pointed out that the compounds of transition element such as manganese with the metalloids are metallic. The metallic condition appears to be connected with the small dimensions of the metalloid atoms. These metallic phases have embedding structures. In such structures the metalloid atoms are always so arranged as to yield a maximum coordination number, hence the face-centered cubic lattice is frequently found. Some compounds of manganese with boron, phosphorus, tin, and arsenic are ferromagnetic. Perrson's (302) suggestion that ferromagnetism is connected with orientation of the manganese atoms in a face-centered structure is in general agreement with these facts.

Manganese-tin compounds have been investigated by Potter (307), who finds that SnMn_4 and SnMn_2 are ferromagnetic. The Curie point of SnMn_4 is 150°C . and the saturation value about 100 units. The Curie point of SnMn_2 is 0° and the saturation value 470 units or virtually that of nickel.

The arsenide and antimonide of manganese have been investigated structurally by DeJong and Willems (65), who find that they have the FeS structure with $a = 37.4$, $c = 5.75$ for MnAs , and $a = 4.14$, $c = 5.90$ for MnSb . Bates (17) has investigated the magnetic properties of the arsenide and phosphide.

The nitrides of manganese have been investigated in some detail by Haag (131). From 0 to 2.2 percent N. the δ phase is stable at high temperatures but decomposes in α manganese and the ϵ phase between $600^\circ - 400^\circ\text{C}$. The manganese atoms in the δ phase are arranged in a face-centered tetragonal lattice with $a = 3.765$, $c = 3.684$. The ϵ phase exists from 6 - 6.65 percent nitrogen at 400°C . In this phase the manganese atoms form a face-centered cubic lattice. The unit cell contains 4 manganese atoms and 1 nitrogen atom. At temperatures above 500°C . the ϵ phase field is enlarged toward smaller nitrogen contents.

The telluride of manganese (MnTe_2) has a pyrite structure $a = 6.943$, according to Oftedal (281).

Manganese forms two sulphides (MnS and MnS_2) known mineralogically as alabandite and hauerite. The monosulphide is particularly important in metallurgy. According to Schneiderhöhn (340), it has a structure analogous to rock salt, with $a = 5.21$. It has a very high index of refraction, $N_{Li} = 2.70$. Its most interesting characteristic is its ability to behave

⁶ Chief engineer, Metallurgical Division, U. S. Bureau of Mines.

both as a metallic and nonmetallic substance. It is, for example, soluble in molten silicates as well as in molten sulphides. The system MnS-MnSiO_3 has already been mentioned. Its equilibrium with iron sulphide has been investigated by Shibata (346)

Manganese sulphide therefore has the unusual property of being soluble in both metallic and nonmetallic melts. The question at once arises as to its distribution between such melts, but no data are available.

Other very peculiar properties are reported for manganese sulphide; thus Picon (305) reports that molten MnS will dissolve graphite. Krekel (216) reports that manganese sulphide and iron form a eutectic.

NONMETALLIC COMPOUNDS AND THEIR USES

Oxides of Manganese

Manganese forms four oxides-- MnO_2 , Mn_2O_3 , Mn_3O_4 , and MnO . The equilibrium relations and natural occurrence of these oxides are discussed elsewhere in this bulletin. Two more states of oxidation are represented in the salts of manganese, MnO_3 and Mn_2O_7 .

The oxide MnO is formed when any of the higher oxides are reduced by hydrogen below $1,200^\circ$, is pale green, and melts at $1,605^\circ\text{C}$. It crystallizes in a cubic lattice of the NaCl type, with $A_0 = 4.40 \text{ \AA}$. MnO is isomorphous with CaO , CdO , MO , and MgO but exhibits only partial solid miscibility with them. Solid MnO dissolves 31 percent MgO and 21 percent CdO , while MgO and CdO dissolve 26 percent MnO . Solid MnO dissolves about 9 percent CaO (269, 270).

The system $\text{MnO} - \text{FeO}$ has metallurgical importance and has been studied by Andrew and Maddocks (9).

The compound MnFe_2O_4 can be prepared by calcining the mixed hydroxides at 800°C . It is cubic, with $a = 8.515$ and a calculated density of 4.965 (Passerini) (294).

MnO is soluble in acids and also in ammonia.

The oxide Mn_3O_4 is formed when the other oxides of manganese are heated in air above 940° . Obtained in this way it is brownish or reddish in color but when occurring naturally as the mineral hausmannite it is black.

Manganic oxide (Mn_2O_3) is formed by igniting the dioxide, nitrate, carbonate, or oxalate below 940°C . It is a brown powder which shows a cubic crystal structure of the corundum type-- $a = 9.391$ (Wretblad) (395). It forms solid solution to a limited extent with Fe_2O_3 . Mn_2O_3 behaves with regard to acids as if it were MnO , MnO_2 .

MnO_2 has only feebly basic properties. When heated with sulphuric acid manganese dioxide evolves oxygen and passes into solution, manganic sulphate being formed. Brown solutions of the disulphate can, however, be formed by

anodic oxidation of manganous sulphate. The tetrachloride is also unstable but may be produced as the double salt $\text{MnCl}_4 \cdot 2 \text{KCl}$ by passing HCl into a solution of KMnO_4 in glacial $\text{HC}_2\text{H}_3\text{O}_2$.

The acidic character of MnO_2 is more pronounced than the basic. When hydrated manganous oxide is oxidized by air in the presence of Ca(OH)_2 calcium manganite is formed.

MnO_2 is readily soluble in acids with the addition of a reducing agent, such as SO_2 . Except as a source of manganese, none of the oxides but the dioxide have any commercial uses.

The principal use of the dioxide is as a depolarizer in dry batteries. For this purpose high-grade MnO_2 is required. Both artificial and natural products are used. Apparently the hydrated form gives better results (193, 238).

Manganese chloride (MnCl_2) is obtained as a byproduct in the preparation of chlorine from MnO_2 . It is isomorphous with FeCl_2 , and its purification by crystallization is therefore impractical.

A mixture of manganous chloride and ammonium difluoride has been recommended by Ferrari (93) for degasifying aluminum (355).

Manganese carbonate (MnCO_3) occurs naturally as rhodochrosite. It is rhombohedral ($a = 5.84$, $\alpha = 47^\circ 20'$) and according to Kreiger (215) forms a continuous series of solutions with calcium carbonate. It is also isomorphous with iron carbonate and magnesium carbonate, and such mixtures occur naturally as oligonite. The carbonate is formed by the addition of the alkali carbonates to a solution of manganous salts.

Manganese Sulphate

This salt is used as a drier in light oils and especially as a component of fertilizers. (Mazzuchelli) (257). The salt crystallizes from water in several hydrated forms. The anhydrous salt takes up water from the air, and the higher hydrates lose water; the monohydrate, however, is stable in air and is therefore best suited to shipping. It is obtained by heating the higher hydrates at $100^\circ\text{--}200^\circ\text{C}$. (Krepelk and Rejha) (217).

Manganates and Permanganates

The higher oxides of manganese - MnO_3 and Mn_2O_7 - are acidic and occur as salts formed by oxidation of manganous salts. When MnO_2 is fused with sodium or potassium carbonate or hydroxide manganates are formed. Manganates form green solutions stable when alkaline but break down when neutral or acid, forming the permanganate and MnO_2 . In commercial preparation of permanganate, oxidation to the heptavalent state may be obtained by chlorine or electrolysis.

Manganese Ores in Ceramic IndustriesGlass Making

Practically all the raw materials used in glass contain some iron, usually in the form of ferric oxide. The iron, when present even in small quantity, imparts to the glass a pale green color that increases rapidly in intensity as the iron content increases. If a colorless glass is desired this green must be removed by some decolorizer. Manganese, selenium, cobalt, and nickel are the most common decolorizers in use, and of these manganese has been most widely employed because it permits easy control of the color. In using selenium and nickel the quantity must be carefully controlled, but these latter substances are desirable, especially in window and plate glass, because glass decolorized with manganese often changes to a pink on exposure to the light. A decreasing quantity of manganese is being used by makers of tank glass, its place being taken by selenium.

The quantity of manganese used varies considerably, depending on the character of the glass, the method of its manufacture, the iron content of the raw materials, and the character of the manganese ore used. Each manufacturer has his own ideas on this subject. The quantity used is figured in terms of pounds of manganese dioxide per thousand pounds of sand, which constitutes 50 to 75 percent by weight of the entire batch. The temperature employed in the glass-making process influences the quantity of manganese dioxide used since, because of volatilization, more manganese is necessary at higher temperatures. The maximum limit is 10 to 15 pounds of manganese dioxide per thousand pounds of sand, and the minimum may be 2 to 2½ pounds.

Chemistry of Use of Manganese in Glass-Making Process. - When other coloring ingredients are absent, compounds of manganese produce pink, purple, and violet hues, according to the chemical nature of the glass. Manganese dioxide neutralizes the green color caused by iron compounds. Used in excess it imparts an amethyst tint, and when used in considerable excess the color is so dark as to appear black.

Neutralization of the iron tint by manganese dioxide is explained by some chemists as a physical effect, and by others as a purely chemical one. The green tint is due to the presence of ferrous silicate. Some chemists think that this green compound is oxidized to the ferric silicate, which is an almost imperceptible pale straw-yellow. According to this view, the oxidizing agent used must not decompose completely at high temperatures, and manganese dioxide seems to be the most available compound fulfilling this condition. At red heat the dioxide loses one third of its oxygen, leaving the tetroxide (Mn_3O_4), which at still higher temperatures is an oxidizing agent.

Red lead and other oxidizing agents have not this decolorizing power, hence some chemists have thought that the decolorization by MnO_2 is purely physical and is not due to oxidation or chemical reaction. It is possible, however, that other compounds may lose their oxygen at too low temperatures to be effective as oxidizing agents.

Specifications for Manganese Ore Used in Making Glass. - Before the World War the ordinary specifications for manganese ore used in glass making were 85 to 90 percent manganese dioxide and less than 1 percent metallic iron. Outside of these two ingredients, each manufacturer has his own requirements. Special glasses may require ore carrying more than 90 percent manganese dioxide and less than 0.5 percent iron. The higher the manganese content and the lower the iron, the better the ore is for glass making. In general, the grades of ore are similar to those used in making dry cells. Obviously, siliceous pyrolusite is not objectionable but carbonaceous pyrolusite is.

Manganese ore for glass making is sold in powdered, granulated, or lump form. There are objections to the lump form because of the time required to melt it into the batch. Powdered ore is used principally when the batch is melted in pots; lump or granular ore is used when melting is done in tanks.

Before the war high-grade pyrolusite for glass making and other chemical purposes was imported from Russia, Saxony, Japan, Nova Scotia, and other foreign countries. As the war progressed such ore became scarce; in consequence, specifications were relaxed, and low-grade ores were purchased. During the war some excellent domestic ore was developed and found a ready market.

Other Ceramic Uses

Another use for manganese ore in glass making has developed in the last few years, namely, for producing black glass used for ornamental purposes. About 3 percent of ore is added to the batch in making this opaque glass.

Pyrolusite is added to the constituents of glazes and enamels to produce purple tints. Black enamels are those containing manganese. Manganese oxide is also used in brick making. Manganese has also been recommended for improving the color of brick (39).

Use of Manganese Salts in Driers

Definition

Driers are substances, generally metallic oxides or their compounds, that are added to linseed or other drying oils at high or low temperatures to make them capable of readily absorbing oxygen from the air, or of drying by its action. Some chemists consider the action to be catalytic, the manganese compound acting as a catalyzer or carrier of oxygen. The principal manganese compounds used as driers are manganese sesquioxide (Mn_2O_3), pyrolusite (MnO_2) --also known in the trade as dioxide, binoxide, or peroxide), manganese hydrate, manganese sulphate, manganese borate, manganese resinate, manganese linoleate, manganese oxalate, and possibly other salts. Certain corresponding double salts of manganese and lead are often used.

Manganese Dioxide

Manganese dioxide, extensively used as a drier, is marketed in two forms, the natural and the artificial. The natural mineral pyrolusite is

simply ground to a powder with water and then dried. The mineral is essentially a peroxide, a class of substances containing more oxygen than is required to satisfy the valence of the metal present. This extra oxygen is loosely combined and readily enters into combination with oxidizable bodies. This feature of manganese compounds makes them useful in oil boiling because the oxygen combines with the oil, oxidizing it, while some of the manganese dissolves and forms a compound with the linoleic acid of the oil. In consequence of this action manganese compounds are powerful driers. The quantity of manganese dioxide added in the process of boiling is small, not more than a quarter of a pound to a hundredweight of oil to get the best results. The use of the black dioxide, however, tends to darken the oil.

Manganese Borate

Manganese borate is perhaps the least objectionable of all manganese salts used as drying agents, although the black oxides are used more. To make the borate, 1 part of manganese sulphate is dissolved in 10 parts of distilled water, and a little of this solution is added to some soda solution to determine whether any iron is present. If pure, the manganese sulphate solution is added to the hot borax solution as long as a precipitate forms. The precipitate is filtered, washed with hot water, and dried.

Manganese Resinate

Resinates, formed by the combination of resin or resin with certain metallic oxides, are much used in making varnish and paint and with the exception of the linoleates are most readily soluble in linseed oil. Manganese resinate is made as follows: Soda ash is dissolved in water, and the solution is boiled with steam; the proper proportion of light coarsely powdered rosin is added, then more soda ash. The clear solution is run into a clear manganese chloride, the resinate separating as a white flocculent precipitate, which is filtered, washed, and dried.

Manganese Linoleate

Manganese linoleate is prepared by pouring a solution of a soap, made by boiling linseed oil and caustic soda, into a solution of manganese chloride or manganese sulphate. The dark-brown plasterlike mass is susceptible to oxidation. When exposed to air the surface becomes covered with a hard, rather insoluble, protective skin. The material should therefore be kept in tightly closed vessels. It acts both as a bleaching agent and a drier for linseed oil. One pound, mixed first with 5 pounds of linseed oil and the whole poured into 10 gallons of linseed oil at 250° F. gives a good drying oil.

The double resinates and linoleates of manganese and lead are also in use.

Manganese Oxalate

Manganese oxalate is prepared by precipitating manganese salts with oxalate of soda or potash or by treating manganese hydrate or carbonate with oxalic acid. One of the advantages of the oxalate is said to be that it decomposes during oil boiling. The manganese dissolves in the oil in combination with linoleic acid, and the oxalic acid is decomposed with evolution of carbonic acid. One fourth to one half pound may be used per hundredweight of oil.

CHAPTER 4. OCCURRENCE AND ORE-DRESSING POSSIBILITIES
OF ORES OF MANGANESE IN THE UNITED STATES

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By Fred D. DeVaney⁷ and Will H. Coghill⁸

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INTRODUCTION

Present-day metallurgical practice requires a large amount of manganese, of which the steel industry is the largest consumer. The chemical and ceramic industries also need manganese-bearing ores. These demands are so great that the consumption of manganese by our industries far exceeds our domestic production. This condition can in part be attributed to the fact that, although large amounts of manganese oxide and carbonate ores occur in the United States, relatively little is of the required grade. Concentration would be necessary. Lack of a sufficient supply of high-grade manganese ore and of well-developed concentration processes places the nation in a vulnerable position when large quantities of manganese must be imported for use in the manufacture of steel. An economical and efficient means of concentrating the low-grade oxide and carbonate ores would augment domestic production and reserves and would be a safeguard in times of emergency.

For about 2 years the United States Bureau of Mines, in cooperation with the Missouri School of Mines and Metallurgy, carried on at its Mississippi Valley Experiment Station, Rolla, Mo., an investigation of the manganese bearing ores of the country. In the course of this research ores from 116 mines or prospects in 20 States were examined and studied to determine their amenability to concentration by gravity processes, such as jigging and tabling, and by flotation and magnetic methods. Leaching processes, which are frequently placed in the ore-dressing category, are discussed in chapter 7.

In addition to determining amenability to treatment by the older well-known methods - gravity and magnetic concentration - an endeavor has been made to improve old methods and to devise means of treating ores that cannot be concentrated by methods now in use.

Outstanding results of this study include the development of flotation processes for concentration of oxide ores and carbonate ores and the application of better gravity-concentration methods in certain more important mining districts. The large amount of data assembled while samples from various districts were studied should prove of value in future, as they indicate the grade of concentrates that can be made.

⁷ Associate Metallurgist, U. S. Bureau of Mines.

⁸ Supervising Engineer, ore-dressing section, Metallurgical Division, U. S. Bureau of Mines.

No attempt has been made to compute concisely the amount of ore in the various deposits or to estimate the cost of concentration processes. The tonnages of ore in many deposits are as yet uncertain and can be determined only by further exploration.

The costs of the processes mentioned are well known throughout the present milling industry, and the average will apply here.

MANGANESE DEPOSITS IN THE UNITED STATES

A survey of the literature shows that manganese ore is known to occur in about 300 districts scattered throughout 35 States. The size of the deposits and the manganese content of the ore vary greatly. Many deposits are small, containing only a few hundred or a few thousand tons, while others are known to contain ore that is measured by the millions of tons. The grade of ores in these deposits varies between wide limits; some contain only a small percentage of manganese, while others are of commercial grade without need of concentration.

A map and bibliography (147) of the manganese deposits of the United States, showing the widespread distribution and also giving a list of the references describing the various districts has been published. This map is reproduced as figure 1.

The ore samples for the investigation were obtained in several ways. Some were taken by Bureau of Mines engineers, and others were sent in by consulting engineers. Many of the samples, however, were supplied by the operators. When the investigation was begun operators were asked to furnish samples. This request met with a ready response. Of course, under some circumstances it would not be good engineering to allow the interested parties to do sampling, but in this instance a true representation of average-grade ore was not required; a knowledge of the general character of the ore was sufficient.

Of the 116 prospects or mines from which ores were examined 20 gave promise of yielding ferro-grade ore, either in the natural state or by concentration, while 43 fell in the class of ores suitable for other metallurgical purposes; thus, 63 properties had ore that was usable. Their phosphorus content would meet specifications, the ratio between manganese and iron was satisfactory, and the silica content was not too high or could be reduced by concentration. If time were available to re-examine these samples in the light of the knowledge acquired throughout the work, without doubt the list of promising ores would be increased. A list of the samples, grouped by States, is given in table 3. It shows how many could be used with or without concentration for the production of ferromanganese, for the manufacture of spiegeleisen, or for addition to the blast-furnace charge to supply manganese to the metal.

The authors do not intend to convey the idea that it would be economical to produce ore of the grade stipulated in the foregoing table from all the ores listed. Many deposits are small and would not justify the erection of a concentrating plant of any sort. Others have a grade of ore that makes them workable only when the price of manganese is high.

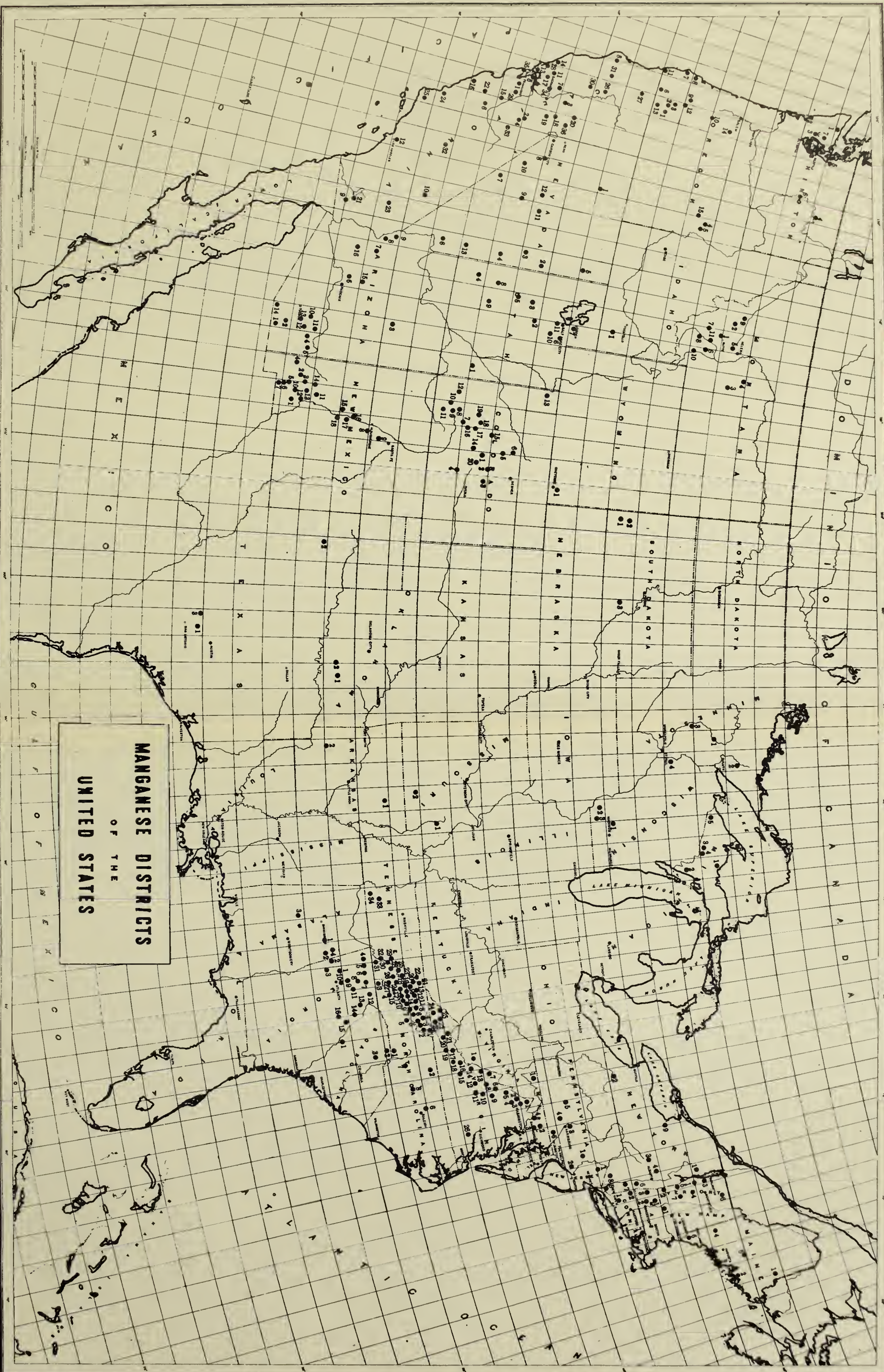


Figure 1.

TABLE 3. - Ores of Manganese Examined, Grouped by States and Grade of Ore Which Can Be Recovered

State	Samples Examined	Ores suitable for production of ferromanganese	Ores suitable for other metallurgical purposes.
Arizona.....	18	4	5
Arkansas.....	22	3	12
California.....	1	-	1
Colorado.....	2	1	1
Georgia.....	1	1	-
Idaho.....	2	-	1
Massachusetts...	1	-	1
Michigan.....	1	-	-
Minnesota.....	7	-	4
Missouri.....	2	-	1
Montana.....	8	3	1
Nevada.....	6	1	3
New Mexico.....	7	1	4
North Carolina..	2	2	-
South Dakota....	1	-	-
Tennessee.....	12	2	4
Utah.....	3	-	1
Virginia.....	6	2	3
Washington.....	13	-	1
West Virginia...	1	-	-
Total.....	116	20	43

District and State:

Kingman, Ariz.
 Batesville, Ark.
 Leadville, Colo.
 Cartersville, Ga.
 Menominee, Mich.
 Cuyuna, Minn.
 Butte, Mont.
 Philipsburg, Mont.
 Franklin Furnace, N. J.

District and State:

Little Florida Mountains, N. Mex.
 Silver City, N. Mex.
 Las Vegas, Nev.
 Chamberlain, S. Dak.
 Northeast Tennessee, Tenn.
 Blue Ridge, Va.
 Woodstock, Va.
 Olympic Peninsula, Wash.

Particular attention has been given to deposits with a record of high production or known to contain relatively large amounts of manganese-bearing material. These deposits are in the above districts, arranged alphabetically by States. Their ore-dressing properties constitute the basis of this report.

Ores from many other districts were studied, but lack of space prevents giving the experimental data on all but those which seem most important at present or which have attracted the most attention. Possibly some deposits not listed in this report will become large producers of manganese in the future; hence the absence of a report on certain districts is not to be considered as indicating an unfavorable attitude. For example, no report is given on the deposits in California, which was the second largest producer of manganese ores during the war years and undoubtedly can again produce much ore under favorable conditions. Due to the recent inactivity of the mines in that State representative samples were difficult to obtain.

CLASSIFICATION OF ORES OF MANGANESE

In this report the classification of ores containing manganese follows the system adopted by the Bureau of Mines in 1913. The term "manganese ore" is used to designate ores containing 35 percent or more manganese. Ores containing 10 to 35 percent are called "ferruginous manganese ores", while those containing 5 to 10 percent manganese are termed "manganiferous iron ores". Ores made up mainly of iron and containing less than 5 percent manganese are not classed as ores of manganese but simply as iron ores. The term "manganiferous silver ores" refers to ores that contain more than 5 percent manganese and enough silver to make them more valuable as a source of silver rather than of manganese. "Manganiferous zinc residuum" is a term applied to a byproduct resulting from the roasting of franklinite concentrates at Franklin Furnace, N. J.; it commonly contains 12 to 14 percent manganese.

PRODUCTION OF DOMESTIC ORES OF MANGANESE

Some idea of the relative importance of the various States as to the type and tonnage of manganese ores produced by each may be had from the figures in table 4, which shows the amounts of ore shipped from 1913 to 1930, inclusive. This table includes the tonnage of both fluxing ore and of the zinc residues produced at Franklin Furnace, N. J. It is felt that both types of material should be included, because the fluxing ore was used primarily for its manganese content, which serves a definite purpose, even though it is not recovered, and because the zinc residue which, though a byproduct from the smelting of zinc ores, is nevertheless the raw material from which most of the country's spiegeleisen is manufactured.

MANGANESE MINERALS

Manganese compounds occur in a variety of mineral forms widely distributed throughout the crust of the earth. From the commercial viewpoint by far the most important manganese minerals are the oxides. The world production of manganese outside of the United States comes almost entirely from the oxides. In this country the supply of the oxides is supplemented by the carbonate mineral rhodochrosite, from the deposits at Butte, Mont.

The manganese minerals (oxides, carbonates, silicates, and sulphides) will be described, after which the mineragraphic identification will be discussed. Table 5 gives the chemical composition and some physical properties of the more common manganese minerals.

TABLE 4. - Ores of Manganese, Including Fluxing Ore and Zinc Residues Shipped from Mines in the United States, 1913-30, Inclusive, in Long Tons¹

State	Ore containing 35 percent or more Mn	Ore containing 10 to 35 percent Mn	Ore containing 5 to 10 percent Mn
Arizona.....	52,971	37,448	21,449
Arkansas.....	61,419	108,479	147
California.....	<u>2/</u> 63,328	359	---
Colorado.....	28,853	694,000	71,426
Georgia.....	<u>2/</u> 47,979	<u>2/</u> 157,326	44,890
Idaho.....	5,537	1,839	---
Michigan.....	---	448,013	413,506
Minnesota.....	---	2,715,446	7,088,693
Montana.....	682,884	61,575	---
Nevada.....	<u>2/</u> 31,618	<u>2/</u> 724,086	7,987
New Jersey ³	---	2,069,000	---
New Mexico.....	22,650	349,479	166,996
Tennessee.....	11,978	<u>2/</u> 4,961	96
Utah.....	<u>2/</u> 10,742	<u>2/</u> 18,523	496
Virginia.....	66,458	51,787	52,400
Washington.....	16,275	---	---
Wisconsin.....	---	---	1,819,172
Undistributed ⁴ .	14,215	12,486	---
Total.....	1,116,707	7,454,855	9,687,258

¹ The production figures are taken largely from records published in Mineral Resources of the United States for the various years.

² Small additional production reported with "Undistributed".

³ Zinc residues, production partly estimated.

⁴ Includes production from States with small production and from States where permission has not been given to publish figures.

I.C. 6768 TABLE 5. - Chemical Composition and Some Physical Properties of More Common Manganese Minerals

Mineral	Composition	Assay, percent Mn	Specific gravity	Hardness	Color	Structure	Streak
Pyrolusite.....	MnO ₂	60 to 63	4.73 to 4.86	2 to 2.5	Steel-gray or iron-black.	Radiating columnar but generally granular massive.	Black
Psilomelane...	4 MnO ₂ Mn, Ba, K, 0.0nH ₂	45 to 60	3.7 to 4.7	5 to 6	Steel-blue to black.	Massive, stalactitic, botryoidal.	Do.
Manganite.....	Mn ₂ O ₃ (H ₂ O)	62.4	4.2 to 4.4	4	Steel-gray or iron-black.	Usually in radiating crystals.	Dark reddish brown.
Hausmannite.....	Mn ₃ O ₄	72.1	4.73 to 4.86	5 to 5.6	Black to brownish black.	Generally crystalline.	Brownish black.
Wad.....	Hydrous mixture of oxides.	Variable.	3 to 4.3	1 to 6	Black or brown.	Amorphous or nodular.	Brown or black.
Alabandite.....	MnS	63.1	3.95	3.5 to 4	Iron-black to brown.	Usually granular massive.	Olive-green.
Rhodochrosite.	MnCO ₃	47.8	3.45 to 3.6	3.5 to 4	Light to dark rose.	Usually cleavable; massive to granular.	Colorless.
Rhodonite.....	MnSiO ₃	41.9	3.4 to 3.68	5.5 to 6.5	Rose	Commonly massive, cleavable to compact in imbedded grains.	Do.
Braunite.	3Mn ₂ O ₃ ·nMnSiO ₂	60 to 69	4.75 to 4.82	6 to 6.5	Black or brownish black.	Commonly in euhedral crystals, granular.	Brownish black.
Bemendite.....	3 MnO ₅ ·H ₂ O7SiO ₂	71.0	3.1	6	Light gray or grayish brown.	Fine grained aggregate of plates or fibers.	Do.
Franklinite.....	(Fe, Zn, Mn) O ₂ (Fe, Mn) ₂ O ₃	10 to 20	5.0 to 5.3	5.5 to 6.5	Iron-black.	Massive, granular.	Reddish brown or black.

Oxide Minerals

The chief oxide minerals are pyrolusite, psilomelane, and manganite. Another oxide, hausmannite, although not uncommon has no economic importance except in two districts studied, the Batesville-Cushman, Ark., and the Olympic Peninsula, Wash. Franklinite is important in the Franklin Furnace, N. J. Wad is not a definite mineral but is a term in common use to describe earthy manganese-bearing material with a high moisture content. It is a mixture of manganese oxides.

Pyrolusite (MnO_2)

Pyrolusite, the manganese dioxide, contains 60 to 63 percent of manganese. Most varieties contain several percent of water. It is a black, opaque mineral with a metallic luster and is frequently soft enough to soil the fingers, the hardness ranging from 2 to 2.5. The specific gravity is 4.73 to 4.86. Pyrolusite often occurs as reniform coatings associated with other manganese minerals. It is sometimes columnar and often granular massive. Pyrolusite frequently occurs as pseudomorphs after the characteristic radiating needles of manganite. It is usually a secondary mineral formed by the dehydration and oxidation of other manganese minerals. Due to its physical structure and chemical properties pyrolusite is the manganese oxide preferred by the dry-battery industry.

Psilomelane ($4 MnO_2 (Mn, Ba, K) 0.nH_2O$)

Psilomelane is an oxide of variable composition containing 45 to 60 percent manganese and like pyrolusite usually contains several percent water. Many specimens contain a rather large percentage of barium; some of the samples examined had as much as 15 percent of barium oxide. It is a hard, black, opaque, amorphous mineral with a dull submetallic luster and conchoidal fracture, commonly found in massive, reniform, stalactitic, or botryoidal forms. Its hardness ranges from 5 to 6 and its specific gravity from 3.7 to 4.7. In concretionary masses common in many deposits, psilomelane displays a marked concentric structure due to alternation of hard and soft layers of the mineral. The varying hardness probably is due to variations in the water content of the psilomelane. When free from other manganese oxides psilomelane can be readily identified by its superior hardness and lack of crystallization.

Manganite ($Mn_2O_3 \cdot H_2O$)

Manganite, a hydrous manganese sesquioxide, contains 62.4 percent manganese and 10.3 percent water. It is a steel-gray to iron-black, opaque mineral of medium hardness (4.0) and a specific gravity of 4.2 to 4.4, commonly found in radiating needle forms or in feltlike masses of fine interlocking needles. It has a dark reddish brown streak and a submetallic luster.

Hausmannite (Mn_3O_4)

Hausmannite, an oxide of manganese, theoretically containing 72.1 percent manganese, is a black to brownish black opaque mineral, commonly crystalline, with a submetallic luster and a specific gravity of 4.73 to 4.86. It has a hardness of 5.0 to 5.5 and a brownish black streak. A fresh fracture usually shows numerous bright, shiny crystal faces.

Franklinite ((Fe, Zn, Mn) $O \cdot (Fe, Mn)_2O_3$)

Franklinite is commonly found only in the deposits of Franklin Furnace, N. J. Its manganese content is variable, ranging from about 10 to 20 percent. It is an iron-black mineral with a metallic luster and reddish brown to black streak. Its specific gravity is 5.0 to 5.2, and its hardness 5.5 to 6.5. It is usually associated with zincite and willemite, and its structure is massive to granular.

Carbonate Minerals

The most common manganese carbonate mineral is rhodochrosite ($MnCO_3$). However, in addition to this mineral there is a series of isomorphous calcium and manganese carbonates with compositions ranging from that of calcite to that of rhodochrosite. Minerals in this series are termed manganocalcites and are not uncommon. Kreiger (215) found that as the mineral grades from calcite to rhodochrosite the indices of refraction and specific gravity increase gradually. He also showed that variation of the isomorphous constituents could be detected by X-ray diffraction study. In addition to the calcite-rhodochrosite series of manganese carbonates a more complex series is found in the ores of the Chamberlain (S. Dak.) district. These ore minerals probably consist of an isomorphous mixture of calcite, magnesite, siderite, and rhodochrosite.

Rhodochrosite ($MnCO_3$)

The manganese carbonate, rhodochrosite, contains 47.8 percent manganese when pure. It is usually light rose, although other shades are not uncommon. Rhodochrosite is a translucent mineral with a vitreous luster and a colorless streak. Its specific gravity is 3.45 to 3.6 and its hardness 3.5 to 4.0. Rhodochrosite often occurs in cleavable massive form with a distinct rhombohedral cleavage. It frequently occurs with rhodonite, from which it usually can be distinguished by the softness of rhodochrosite and by the reaction of rhodochrosite with warm hydrochloric acid. Chemical analyses or specific-gravity determinations are sometimes necessary to distinguish rhodochrosite from the manganosilicates.

Silicate Minerals

The manganese silicates have no importance as ores of manganese but are of interest due to their association with the oxides. Rhodonite and braunite are the two most common silicates. The hydrous silicate, bementite, is found

in large quantities in the manganese deposits of the Olympic Peninsula, Wash., where it sometimes is the major constituent of the ore body. Occasionally neotocite, a rather rare mineral of similar composition, is found with the bementite in the Washington deposits.

Rhodonite (MnSiO_3)

Rhodonite, the metasilicate of manganese, theoretically contains 41.9 percent manganese and 45.99 percent silica. It is usually deep rose and translucent, with a vitreous luster and a colorless streak. The fracture of rhodonite is conchoidal to uneven. Its specific gravity is 3.4 to 3.7, and its hardness (5.5 to 6.5) is sufficient to distinguish it from rhodochrosite, with which it is sometimes confused. Its structure is commonly massive, cleavable to compact, or in embedded grains.

Braunite ($3 \text{ Mn}_2\text{O}_3 \cdot n \text{ MnSiO}_2$)

The silica content of braunite is variable, and according to Schneiderhohn and Ramdohr (³⁴¹) the silicate molecule may sometimes be absent. Most varieties contain about 60 percent manganese and about 9 percent silica. Braunite is a black to steel-gray opaque mineral with a submetallic luster and a brownish black streak. Its specific gravity ranges from 4.72 to 4.82 and its hardness from 6.0 to 6.5. It commonly occurs in granular form with euhedral tetragonal crystals. The crystallizing tendency of braunite is so well pronounced that it usually serves as a method of identification.

Bementite ($8 \text{ MnO} \cdot 5 \text{ H}_2\text{O} \cdot 7 \text{ SiO}_2$)

Bementite is a hydrous manganese silicate of somewhat uncertain composition, containing usually about 31 percent manganese and 35 percent silica. The mineral from the Olympic Peninsula district is light gray to grayish brown and has a vitreous luster, a splintery fracture, a hardness of about 6.0, and a specific gravity of about 3.1. It is transparent in splinters but upon weathering becomes opaque. Microscopic examination shows that the mineral usually consists of a fine-grained felted aggregate of fibers and plates.

Sulphide Minerals

Two manganese sulphide minerals are known - alabandite (MnS) and hauerite (MnS_2). They have no commercial importance as a source of manganese and are of interest only from a scientific viewpoint. Alabandite is the more common mineral and has been reported by Hewett and Rove (¹⁶¹) as occurring in a number of western localities. Hauerite is a rare mineral.

Alabandite (MnS)

Alabandite theoretically contains 63.1 percent manganese. It is an iron-black to brown mineral with a submetallic luster and an olive-green streak, which is quite distinctive and often serves as a means of identification. Alabandite has a specific gravity of 3.95 and a hardness of 3.5 to 4. It is usually granular massive in structure, with perfect cubic cleavage.

Mineragraphic Identification of Opaque Manganese Minerals

Identification of the various oxide minerals is frequently difficult. Their intimate association and the alteration of one mineral to another are confusing. Pseudomorphs are abundant and misleading. Sometimes positive identification is possible only by study of polished sections under the microscope, accompanied by microchemical tests. Microscopic methods of identification of some of the more important opaque manganese minerals have been given by Thiel (367,368), Short (348) and Fairbanks (89) and by Cooke and associates (58).

The authors have used polarized light in conjunction with the metallographic microscope in the differentiation of the opaque manganese minerals and have found it of primary importance. An examination by this method makes it possible to place them immediately in one of two classes, anisotropic or isotropic.

Anisotropic Minerals.

Three of the more common opaque manganese minerals -- pyrolusite, manganite, and hausmannite -- are strongly anisotropic, each giving four extinctions per revolution. (Most anisotropic minerals give only two extinctions.) If these three minerals occur in particles of sufficient size a hardness test suffices to identify them. Pyrolusite is soft (2 to 2.5), manganite is of medium hardness (4.0), and hausmannite is hard (5 to 5.6). In fine-grained aggregates pyrolusite and manganite are not identifiable by hardness tests, and recourse must be had to an etching treatment. A short etch with an 8 percent solution of sulphurous acid rapidly darkens and pits pyrolusite, while manganite is virtually unaffected. Photomicrographs of unetched and etched sections of pyrolusite and manganite are shown in figure 2, A and B. Hausmannite may be differentiated from pyrolusite and manganite by its hardness, by the excellent polish it takes, and by a characteristic reticulate etch pattern with a saturated solution of stannous chloride. Figure 3, A, shows the usual crystalline structure of hausmannite and the strong anisotropism of the mineral.

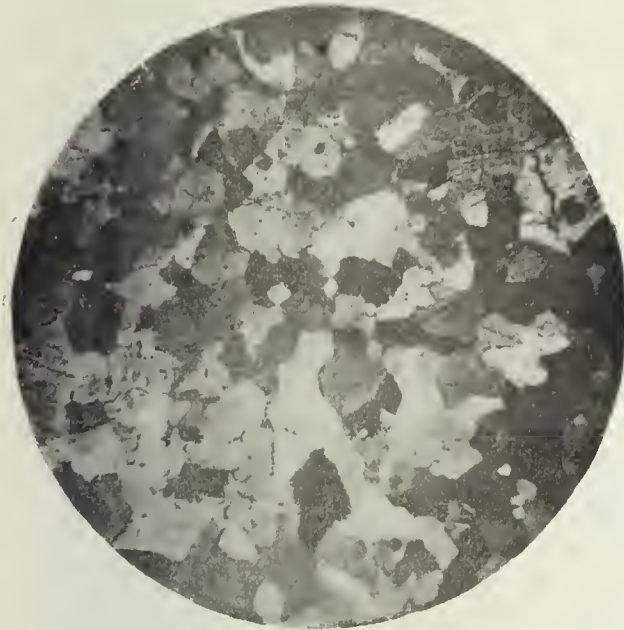
Isotropic Minerals.

Four of the more common opaque minerals are isotropic -- psilomelane, braunite, franklinite, and alabandite. Psilomelane is amorphous and when etched with sulphurous acid darkens, giving a sorbitic structure. Banded structures are rather common. Psilomelane of several generations is often present in the same specimen. Figure 4, A, shows a specimen of psilomelane unetched, the secondary psilomelane being much clearer than the older variety. Figure 4, B, shows the large amount of quartz in the primary psilomelane and the sorbitic pattern brought out by etching. Braunite usually can be identified by its tendency to form euhedral crystals and by the fact that it is not etched by sulphurous acid and but slightly by stannous chloride. The twinning sometimes shown by braunite is brought out in figure 3, B. Franklinite occurs commonly only in the mines at Franklin Furnace, N. J. A blowpipe test on charcoal for zinc is the surest means of differentiating it from the other manganese minerals. Alabandite can be readily identified by its olive-green streak and

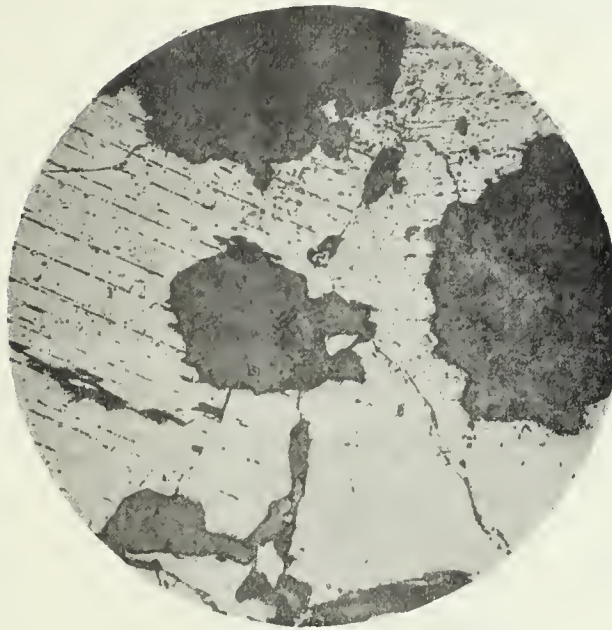


B

Figure 2.—Pyrolusite and manganite. Little Florida Mountains, N. Mex. Dark areas pyrolusite, light areas manganite. (75 x). A, Unetched. B, Etched 5 seconds with 8 percent solution of sulphurous acid.

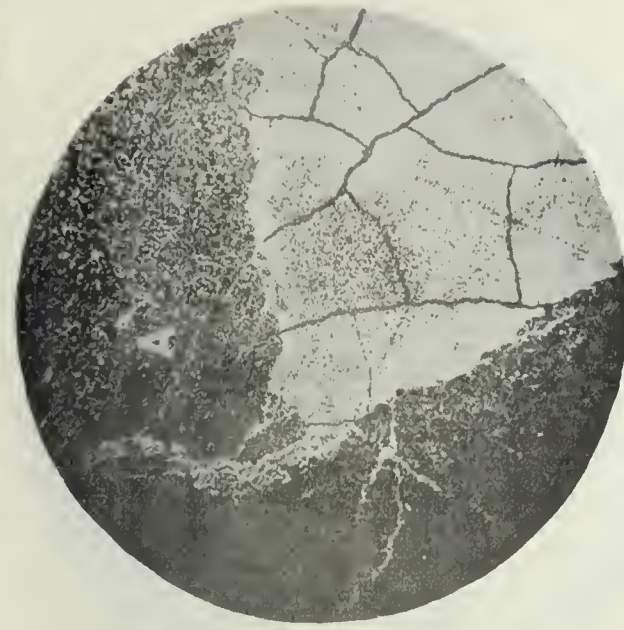


A

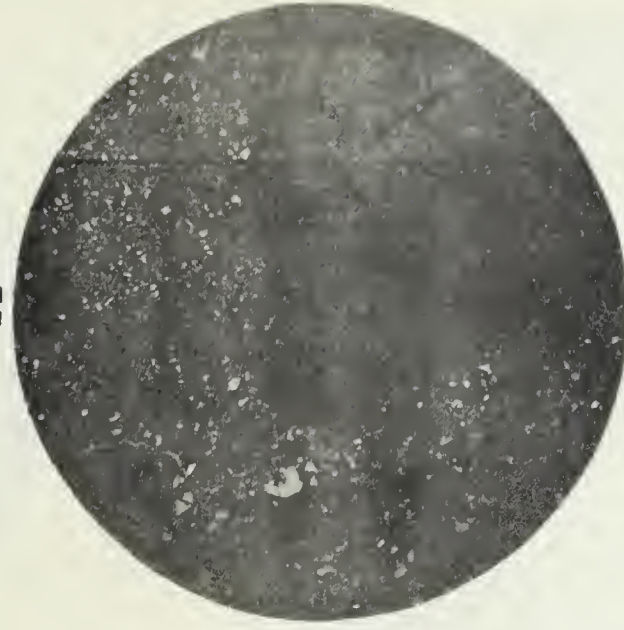


B

Figure 3.—A, Hausmannite. Cushman, Ark. Polarized light. Unetched, 70 x. B, Braunite. Light area braunite; dark area gangue. Note twinning in braunite. Unetched, 75 x.



A



B

Figure 4.—Psi-lomelane. Crimora mine, Virginia. Bright areas are quartz. (70 x). A, Unetched; (1) pure psi-lomelane; (2) secondary. B, Etched with 8 percent sulphurous acid for 90 seconds.

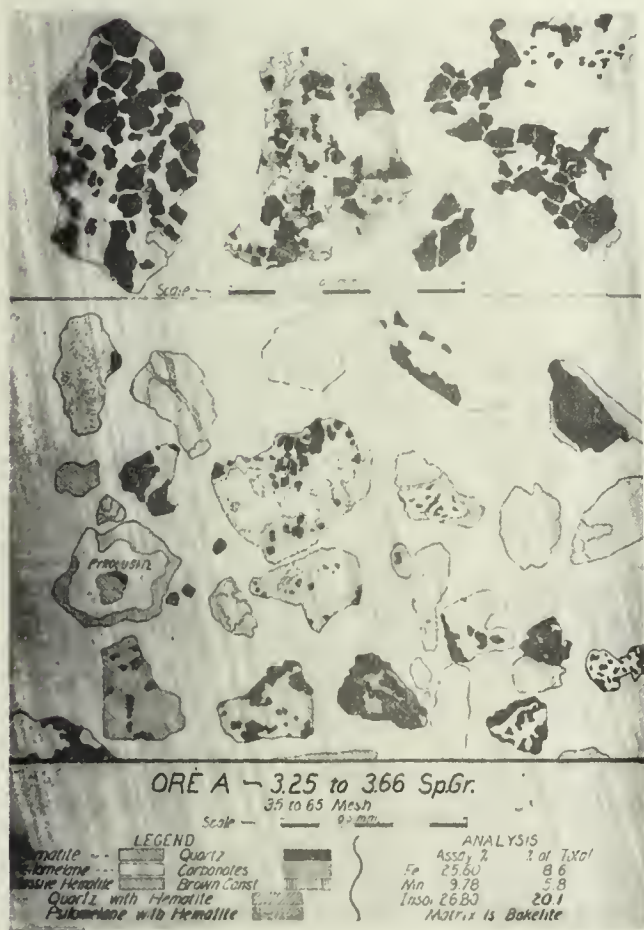
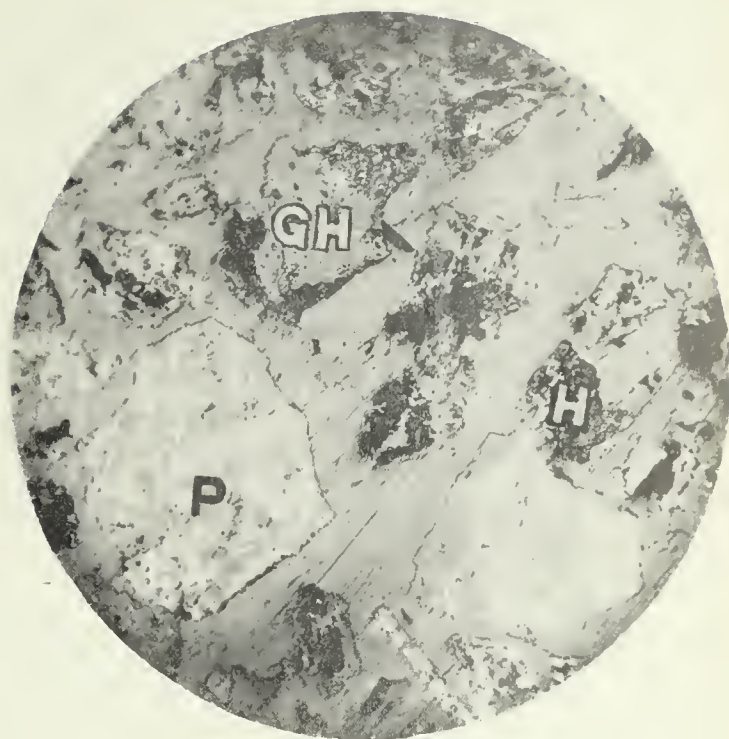
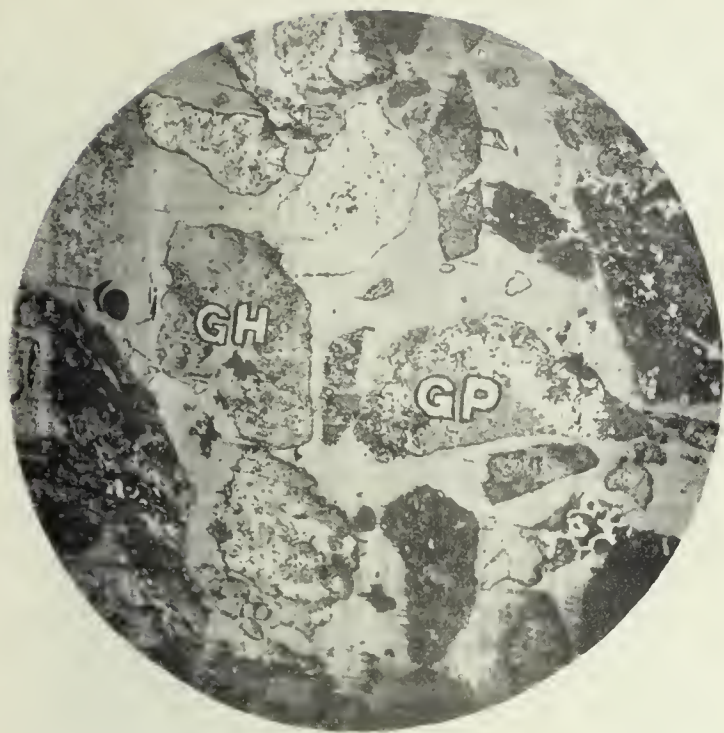


Figure 5.- Briquetted Cuyuna ore.
A, Polished section; 35- to 65-mesh portion, float-and-sink fraction 3.25 to 3.66 sp. gr. Illustrates intimate interlocking of siliceous gangue (G), hematite (H), and psilomelane (P). 50 x. B, Camera lucida drawing of selected particles.

Figure 6.- Briquetted Cuyuna ore..
A, Polished section; 35- to 65-mesh portion, float-and-sink fraction 3.25 to 3.66 sp. gr. Gangue (G), hematite (H), psilomelane (P). 50 x. B, Camera lucida drawing of selected particles.

TABLE 6. - Mineralogical Characteristics of More Common Opaque Manganese Minerals

Mineral	Examination in polarized light	Hardness	Color	Color of Powder	Structure	Reaction to etching solution		
						8 percent solution of H_2SO_4	Commercial H_2O_2	Saturated sol. $SnCl_2$
Psilomelane.....	Isotropic	5 to 6	Gray.	Black.	Nodular amorphous.	Darkens with sorbitic pattern.	Effervesces vigorously with etching.	Tarnishes with etching.
Braunite..	do.	6 to 6.5	do.	Brownish black.	Commonly granular with tetragonal crystal outlines.	No reaction.	Slowly effervesces without staining.	Etches slightly to grayish brown.
Alabandite	do.	3.5 to 4	Sphalerite gray.	Olive green.	Usually granular massive, perfect cubic cleavage.	Etches rapidly; stains white; H_2S liberated.	No reaction.	Tumes tarnish; H_2S liberated.
Pyrolusite	Anisotropic.	2 to 2.5	Gray, almost white.	Black.	Radiating columnar and granular massive; frequently pseudomorph after manganite.	Darkens rapidly; deeply etched.	Effervesces vigorously.	Deep etch; pitted.
Manganite.	do.	4	Gray.	Brownish black.	Usually in radiating crystals.	Darkens slightly with long etch.	Slowly effervesces without staining.	Short contact brings out crystal structure.
Hausmannite	do.	5 to 5.6	Bluish white.	Dark reddish brown.	Generally crystalline.	No reaction.	do.	Tarnishes brown; rubs to reticulate etch pattern.

by the liberation of hydrogen sulphide when etched with stannous chloride. Table 6 shows the differentiation of the more common opaque manganese minerals. For the transparent minerals ordinary petrographic methods are used.

Impurities Associated with Manganese Minerals

Manganese ores, like those of other metals, contain impurities which must be removed by ore-dressing or metallurgical processes. Some impurities are present as discrete particles, which can usually be rejected by ore-dressing methods unless too finely intermixed. Chemically combined or dissolved impurities can be eliminated only by metallurgical processes.

Some ores containing manganese are mined in the West for a flux in non-ferrous metallurgy. Frequently they contain, besides manganese, some lead, zinc, silver, gold, and other metals. Because no manganese is recovered from concentration of these ores they will not be discussed.

The impurities commonly found in ores of manganese may be classified into four general groups, namely:

(1) Metallic--iron, zinc, lead, copper and silver; (2) gangue--silica, alumina, lime, magnesia, and baryta; (3) volatile--water, carbon dioxide, and organic matter; and (4) miscellaneous--phosphorus and sulphur.

Of the metallic impurities, iron is by far the most common. The iron occurs in oxide form, which is usually difficult to remove by ore-dressing methods; magnetic separation may be possible after a reducing roast. Gravity concentration sometimes makes a concentrate enriched in manganese and impoverished in iron. The amount of iron present in a final product determines the type of alloy that can be made. Usually products with more than 1 part of iron to 9 parts of manganese cannot be used for the manufacture of ferromanganese. The other metallics, such as zinc, lead, copper, and silver, in ores utilized for their manganese are found in merchantable quantities in the Butte carbonate deposits. With the exception of silver, these elements are present as sulphides, and although they occur in rather small quantities they should be removed by ore-dressing methods. The chief reasons for removal are that sulphur and zinc are objectionable in the metallurgical processes for the production of ferro-alloys and that economic losses result because none of the metals are recoverable in these processes.

Gangue, such as silica, alumina, lime, magnesia, and baryta, is found in ores of manganese. These impurities, which comprise the slag-forming constituents of the ore, must be kept within allowable limits if the metallurgical treatment be economic. They pass largely into the slag when smelted, and although a certain amount of slag is essential for the removal of injurious impurities, such as sulphur, a large amount of gangue with the consequent production of large slag volumes is highly undesirable. Large amounts of gangue require large amounts of coke. Much slag is formed, and as the slag always contains manganese, the amount of metal recoverable, other conditions being equal, is inversely proportional to the amount of slag formed.

The gangue constituents frequently are classed as acid or basic, according to their behavior in the furnace. Of the more common impurities silica and alumina are usually considered as acid, although the action of alumina is somewhat uncertain. Lime, magnesia, and baryta are considered as basic. Manganese-alloy furnaces are run to produce basic slags and for this reason basic impurities are less objectionable than acid impurities. Silica is by far the most common of all the gangue and is found in nearly all ores. The elimination of silica is usually the chief problem confronting the ore-dressing engineer. Alumina in the form of clay is a troublesome impurity in many ores. When it occurs with hard or nodular manganese minerals its elimination is usually easy, but when the manganese minerals are soft and friable removal is difficult. The basic impurities lime, magnesia, and baryta are seldom found in troublesome quantities. However, in some districts where the manganese mineral is predominately psilomelane, undesirable amounts of baryta may be present. In such instances the barium content of the ore cannot be lowered by ore-dressing methods. In the Chamberlain (S. Dak.) district lime and magnesia are present as carbonates in an isomorphous mixture with rhodochrosite and siderite. Their elimination by ore-dressing methods does not seem probable. With the exception of the Chamberlain ore, or ores of psilomelane with high barium content, the basic impurities in the ores studied have been present in small quantities or can be easily rejected.

Water and carbon dioxide are the two important volatile impurities. Water is present to some degree in virtually all ores. It is not simply a diluent but also lowers the metallurgical efficiency of the blast furnace. The removal of water is frequently desirable. Some wad ores of Arkansas contain as much as 30 percent of water, and since these ores are usually marketed in the Birmingham district the fallacy of paying freight on so much water across several States can be readily seen. Carbon dioxide is important only in rhodochrosite ores. According to some blast-furnace operators carbon dioxide is harmful to the operation of a furnace, as it interferes with the usual carbon monoxide: carbon dioxide ratio. Whether or not this difficulty exists it is important that rhodochrosite ores be calcined before shipment, as the manganese content can be increased materially and freight charges saved by elimination of the carbon dioxide, which in pure rhodochrosite amounts to 38.3 percent.

Sulphur and phosphorus are objectionable impurities in any manganese ore. Upon smelting, the phosphorus content of the ore passes almost entirely into the ferro-alloys. Since low-phosphorus alloys are required, a high-phosphorus ore cannot be used for making ferromanganese or speigeleisen. Phosphorus is usually chemically combined or so intimately associated with the manganese and iron that its elimination by ore-dressing methods is impossible.

In smelting operations sulphur usually passes into the slag, hence its effects are not as harmful as those of phosphorus. Sulphur was not found to any extent in domestic ores, except in the Butte carbonate ores, where it occurs as sulphides. The sulphides usually can be eliminated by ore-dressing processes.

ORE DRESSING OF ORES OF MANGANESE

Ore dressing is the art of physical separation of ore from gangue. The segregated ore is "concentrate" and the gangue is "tailing". To effect a separation of ore and gangue various physical properties of the minerals are utilized. The use of these physical properties to segregate discrete mineral grains differentiates ore dressing from metallurgy, which deals with chemical reactions. In metallurgy some natural compounds are destroyed for new ones more favorable to the separation sought. Ore dressing permits removal of some of the gangue before metallurgy begins; thus, it is a link between the mine and the metallurgical plant.

Ore dressing hypothecates the liberation of discrete economic mineral grains; the gangue mineral grains need not be liberated from each other. The first step in liberation is incident to the mining, as a result of blasting. After that breaking, crushing, and grinding follow until liberation is obtained.

The more important physical properties employed in dressing ores of manganese are color, luster, hardness, magnetic permeability, specific gravity, and surface phenomena.

Color and Luster

Hand-sorting or picking may be employed when the minerals are liberated from each other, and the color and luster permit identification. Sometimes the mineral is removed by hand, sometimes the waste is removed, and sometimes each is thrown into its respective bin. The mineral pieces removed are 2 inches to 2 feet in size. The required labor is cheap, and the skill in close differentiation by trained men and women sorters puts to shame the best college mineralogists. One advantage of hand-sorting is the removal of rope, wood, steel, and dynamite incident to mining.

Hardness

With respect to the oxide ores of manganese, with the possible exception of pyrolusite, a rule about hardness may be laid down, namely, that the coarsest sizes (the hardest) resulting from crushing have the highest manganese content. Sometimes screening on a single screen of the correct size will make concentrate and tailing. At any rate, the hardness characteristic favors gravity concentration, which requires material that is not too fine. This hardness characteristic warrants further comment. Not only are the coarser sizes of higher manganese content than the finer sizes, but also the coarse manganese-bearing grains are richer than the fine manganese grains. To state this relationship in another way: The hardness increases with the purity. This is not true of rhodochrosite, which occurs in a high state of purity and has clean mineral in all sizes. On crushing rhodochrosite fine enough to liberate adhering gangue all sizes contain clean mineral.

Magnetic Permeability

Probably no other ores are so generally permeable to magnetic forces as the ores of manganese. The magnetic permeability of manganese minerals in many instances is due to the presence of iron. Manganese minerals of the highest purity can however be concentrated by the use of high-intensity separators, and the manganese compounds themselves may under certain conditions be ferromagnetic. Rhodochrosite is so susceptible that the commercial use of magnetic separators has been considered. The magnetic separation of an oxide ore, largely pyrolusite, is in commercial practice. High-intensity magnetic separators are required for separation of the manganese minerals. However, the permeability of these minerals is frequently higher than that of the associated iron minerals. Such oxides as pyrolusite and psilomelane have been separated from the less-permeable limonite. Martin (243) states that precipitated manganese dioxide is quite magnetic. The iron content was only 0.096 percent.

The use of the magnetic properties of the manganese minerals as a means of making commercial separations has not been adopted more widely because of the cost of the equipment and the necessity for preliminary drying and sizing of the ore.

Specific Gravity

When the ratio of weights (in water) of unit volumes of gangue and mineral is 1 : 1-1/2 or greater gravity concentration must be considered.

The outstanding methods of gravity concentration are jigging and tabling; the first is by vertical currents of water and the second by horizontal currents. Since these methods, with all the supplementary devices and procedures, are described in the many textbooks on ore dressing and in technical publications the discussion will be abridged.

Jigs and jigging permit many classifications. Only two will be mentioned; the first relates to one of the mechanical features, and the second depends on preparation of the feed. The first is subject to two divisions; the sieve may be movable or it may be fixed.

(1) When it is movable the jig may be called a basket jig because the sieve constitutes the bottom of a frame or basket. The basket works in a tank of standing water and hence is most desirable when water is scarce. The simplest pattern is worked by hand and is called a hand jig. The feed and discharge are intermittent. Its simplicity and low cost make the hand jig a desirable concentrator where a small mine is being developed. Larger basket jigs are driven by power; and some of them, in addition to the up and down motion of the basket, impart a horizontal throwing action which speeds up the travel of the charge. They are at their best in handling coarse material.

(2) Fixed-sieve jigs are more common and are usually home-made. The motion of the water is actuated by a plunger, either at the side of the jig cell or under it. Fresh water must be added to replace what passes over the tailboard and through the hutch. Since it is always added to the plunger the amount affects the rate of suction. When much water is used fine material rides on the charge; when little water is added a suction is created that draws

finer into the hutch. A jigman prides himself on the adjustment of suction to suit the particular ore. Manganese ores, usually lean in the fines, would require minimum suction--that is, maximum amount of water to throw the fines into the tailings.

The other classification--that depending on preparation of the feed--is no less important than the first. The feed may be sized or mixed (unsized).

The sized feed is the only kind that is metallurgically correct. When the ore is difficult--that is, when the mineral is only moderately heavier than the gangue--the feed should be sized between limiting screens with openings that have a ratio not greater than 1 : 1 1/2. To do otherwise is to violate natural laws for expediency. When the mineral is heavy, like galena for example, and liberated from the gangue almost any procedure will recover it, but if it is disseminated through the gangue so that the composite grains have an intermediate density thought must be given to precision. The jigging of manganese ores is most satisfactory when the feed is sized. The best method to draw concentrates is through correctly spaced holes in the sieve.

Table concentration is for material finer than 3 mm. Just as a jig feed requires sizing, so does a table feed require classification with a multiple-spigot hydraulic classifier. The more difficult ores need to be classified more closely than the easy ones. Since the fine sizes of manganese oxides have the low-grade and lightweight grains segregated therein close attention should be given to table concentration.

Before resorting to jigging and tabling the ore should be fractioned on heavy liquids. If heavy liquids do not yield concentrates that are up to grade it is folly to try to make machinery do it. Some manganese oxides are too inherently foul to yield concentrates by any kind of physical separation. Many examples of float-and-sink tests are given in this report.

Surface Phenomena

The practice of flotation began in Australia about 1904 and was taken up with vigor in the United States about 10 years later. It depends on phenomena exhibited at the mineral-water interface. When the ore is crushed fine enough these phenomena predominate over those due to mass, so that minerals with different surface properties may be separated by flotation. Flotation is fundamentally gravity concentration because it depends on the attachment of air bubbles to the mineral grains to make a composite aggregate lighter than water; thus the mineral is brought to the surface of the pulp and removed.

Only during the last decade has flotation of minerals other than sulphides, native metals, and a few of the oxidized copper and lead minerals become important. Many nonsulphide (53) minerals are now known to be floatable, and the number that can be separated by flotation is growing rapidly.

Flotation as a process for the treatment of ores of manganese was investigated, and many ores were found to be amenable to it. Of the manganese minerals, rhodochrosite is readily floatable. The oxide minerals, such as pyrolusite, manganite, and psilomelane, are less floatable, but in many instances satisfactory separation of these minerals is possible. The process works best when the manganese minerals are clean, and the gangue is hard and noncalcareous. The reagents used in the flotation of manganese ores are similar to those used in other nonsulphide separations, namely, soap or a fatty acid as a collector, pine oil or cresylic acid as a frother, sodium silicate as a dispersing agent, and sodium carbonate to control the alkalinity. The flotation of manganese ores other than rhodochrosite has not been demonstrated successfully in large-scale mill operation.

Much literature on flotation is available, and what follows in this report will show some preliminary work on the flotation of ores of manganese.

EXPERIMENTAL STUDY OF ORE DRESSING OF ORES OF MANGANESE FROM VARIOUS DISTRICTS

Scope of Work

The following pages will be devoted to brief descriptions of some of the various manganese districts and to a resume of experimental work at the United States Bureau of Mines Mississippi Valley Experiment Station, Rolla, Mo. The order of treatment is alphabetical by States.

Heavy liquids were used to determine the amenability of the ore to gravity concentration. The rapidity with which tests can be made with heavy liquids made possible the examination of a great many samples. The results of the float-and-sink tests indicate the theoretical grade of concentrates and the recovery of mineral that can be expected from commercial equipment when working on the same size of ore. As a general rule, it may be said that jigs and tables will not give as high a recovery as float-and-sink tests indicated, but this is frequently compensated by a somewhat higher grade concentrate due to the loss of some of the middling grains in the tailing. The use and care of heavy liquids have been reported in other publications (52, 283, 365).

The authors are not in a position to estimate the amount of ore in the various districts, and no attempt will be made to give probable treatment costs of these ores by different processes. The purpose of the investigation was to determine what could be done toward concentrating the manganese ores of the country by ore-dressing methods. This aim has been kept in mind, and the determination of what could be done has been given more thought than the question as to whether such processes would be profitable at present.

Kingman District, Ariz.

Location and Extent

Large amounts of low-grade ores have been reported in recent years as occurring in the district north of the Williams River in Mohave County, Ariz., between Kingman and Wickenburg and known as the Kingman district. These deposits have been described by Jones and Ransome (182) and also by Wilson and Butler (393).

Mineralogy

The manganese in high-grade ore is in the form of psilomelane and pyrolusite, replacing limestone. The manganese minerals are in slaggy vesicular aggregates. The only gangue minerals noted are gypsum, clay, and a small amount of chalcedony.

The samples of low-grade ores examined contained quartz and feldspar grains with some clay. Manganese oxides are scattered in finely divided form throughout.

Concentration Tests

High-Grade Ore. - The sample studied analyzed 38.5 percent manganese, 1.6 percent iron, 16.7 percent insoluble⁷, and 0.025 percent phosphorus.

A slight enrichment can be accomplished by crushing and washing out the slime, which is mostly clay. Float-and-sink tests of ore sized between 1/2 inch and 14 mesh showed no liberation of gangue. Similar tests of ore sized between 14 and 65 mesh showed that the percentage of insoluble⁷ would be reduced from 17.7 to about 15.2 percent if tabling were practiced. Further crushing would undoubtedly favor production of somewhat higher-grade concentrates. The high-grade ores of this district, while limited in amount and quite siliceous, have the desirable characteristics of possessing a low iron and phosphorus content.

Low-Grade Ores. - Three samples of low-grade ores, said to be representative of large deposits, were examined. The samples looked much alike and consisted of brownish black porous material, the particles of which could be readily broken with the fingers. Many pieces glistened, due to the presence of small quantities of calcite. The major part of the material is quartz, feldspar, and clay. Manganese oxides are scattered in finely divided form throughout. Analyses of the samples examined follow:

Sample	Assay, percent				
	Mn	Fe	Insol.	P	CaO
1	8.1	2.3	68.6	0.057	1.69
2	5.7	2.8	72.4	.059	---
3	5.9	2.4	63.6	.059	---

As all three samples had quite similar characteristics, that of highest grade was chosen for experimental work. When the sample was crushed to one-half inch it gave a slight enrichment of the coarser sizes, due to the tendency of the clayey portion to slime.

⁷ To simplify the chemical work many samples listed are reported in terms of insoluble rather than as silica. On most ores the values for insoluble and silica are very close. Insolubles are obtained by dissolving in aqua regia and taking to dryness. The residue is then treated with concentrated hydrochloric acid, diluted with water, and filtered. The insoluble residue and filter paper are ignited and weighed and reported as insoluble. True silica may be determined from the insoluble, if desired, by the usual methods.

Float-and-sink tests of sized portions were made in a liquid with a specific gravity of 2.90 and showed conclusively that high-grade concentrates could not be made by jigging or tabling. The cleanest concentrate contained only 25 percent of manganese and had 38 percent of insoluble. The recoveries of manganese in gravity concentrates would be poor.

There were attempts to concentrate by flotation, but the results were unsatisfactory. Concentrates were made, but their silica content was always high. Intimate association of the mineral with the gangue, large quantities of soluble salts in the ore, and the presence of much clay contributed to the failure of this process.

From the experimental work on the low-grade ore its concentration by present ore-dressing methods appears hopeless. Claims have been made that the ore is amenable to leaching processes. If a low-cost process can be used the deposits may prove valuable. Undoubtedly some higher-grade portions of the ore deposits can be used for metallurgical purposes.

Batesville District, Ark.

Location and Extent

The Batesville district, near the town of the same name in northern Arkansas, has long produced manganese ore and ferruginous manganese ores. This district is an east-west belt 4 to 8 miles in width and about 24 miles in length. The area has particular interest due to the widespread occurrence of manganese minerals and to the variety of the types of deposits in which they occur.

Types of Deposits

Several excellent geological reports are available, the most recent being by Miser (260). The deposits may be grouped in the order of commercial importance as follows:

1. Residual deposits derived from the Fernvale limestone.
2. Deposits in the Cason shale and its residual clay.
3. Replacement deposits in the Fernvale limestone.
4. Replacement deposits in clay.
5. Transported deposits in stream gravels.

A small production has come from the last two types of deposits, but as they have interest only from a scientific viewpoint rather than as a source of supply of manganese they will not be discussed.

Residual Deposits Derived from Fernvale Limestone. - Ore deposits of this type are the most important in the district. They have yielded the greater part of the high-grade ore as well as a considerable proportion of the low-grade ore. The ore occurs disseminated in the residual clay in all gradations of form from large lumps of hard ore to irregular bodies of wad ores. The

chief manganese minerals are psilomelane, hausmannite, wad, and braunite. The latter is the least common. The subject will be discussed under lump ores, manganese-bearing clays, and wad ores.

Lump ore. - The lump ore is of high grade and can be recovered readily from the clay and particles of chert. Some analyses of lump ores mined during 1929 showed that they contain 46.0 to 58.0 percent of manganese and less than 5 percent of silica. These hard ores are also characterized by their low phosphorus content compared with the softer wad ores.

The coarser ores are generally hand-picked, but where the ore occurs in finer form log washing, followed in some instances by jigging, has been found satisfactory. The problem of working these deposits has been largely one of mining. The manganese mineral is irregularly distributed through the clay in pockets surrounded by clay containing little or no ore. Numerous attempts to open mines on a large scale and to employ steam shovels have failed due to the spotty character of the ore deposits, to the large amount of valueless gangue that must be handled, and to the occurrence of limestone pinnacles. Production now comes from a number of small operations where the principal equipment is still wheelbarrows, picks, and shovels.

Manganese-bearing clays. - The recovery of the small particles of manganese minerals associated with clay in the residual deposits is difficult. The amount of manganese occurring in finely disseminated form in the clay is large, and a mill was erected on Folk Bayou by Miller & Rinehart in 1926 to treat this type of material. This operation represented the first definite effort in the district to recover such material.

Methods at this mine were described by Furness (104, pp. 179-180) as follows:

".....The property is equipped with an experimental mill, which consists of two 35-foot log washers, Oliver classifier, Wilfley tables, and a 35-foot rotary drier. The method of mining the ore consists of a partial stripping with a steam shovel, followed by pick, shovel, and wheelbarrow work. The surface of the bedrock limestone on which the residual clays are found is very irregular with extensive solution channels and potholes among residual pinnacles of limestone. The clay containing manganese is trammed to the mill and delivered to the log washers. There is no overflow from the log washers; they are used solely as agitators. The discharge from the log washers, including the wash-water sand and ore, is passed over a fine screen; the undersize from the screen is flushed to Dorr grade C classifiers. The product from these classifiers is treated on Wilfley tables. The oversize from the first screening is crushed by rolls to 10 mesh and then concentrated on the Wilfley tables. The concentrates are passed through the rotary drier where the moisture is reduced to less than 7 percent. The capacity of the mill is about 100 cubic yards per 12 hours, and the recovery is 20 tons of concentrate of approximately the following analysis: Mn, 20 to 25 percent; Fe, 18 to 20 percent; SiO_2 , 14 percent; Al_2O_3 , 12 percent. The rejects and tailings from the tables,

containing from 16 to 18 percent of manganese, are impounded in a tailing pond. The crude ore contains about 12 percent of manganese. The product of the mill is being purchased by the Tennessee Coal, Iron & Railroad Co. at 42 cents a unit. Miller & Rinehart control more than 1,000 acres of land on Skunk Creek, and they claim that the results of drilling and test stripping indicate several million tons of concentrating ore."

The operation of this mill was not satisfactory due to the high insoluble and phosphorus content of the concentrate and the large losses of manganese in the tailings. A sample of the sinter produced from the concentrates had the following analysis: manganese, 17.4 percent; iron, 19.2 percent; insoluble, 20.8 percent; and phosphorus, 1.443 percent.

Concentration tests. - Samples of the manganiferous clay at the Miller-Rinehart property were taken in 1929 and studied to determine the character of the ore and the possibility of making higher-grade concentrates than had been produced. The samples had a tenor of only 6.6 percent manganese, a figure somewhat lower than the reported feed to the mill. Nevertheless, the samples are believed to be fairly indicative of the grade of remaining ore. The ore consists of fine particles of hard manganese oxides and of soft amorphous particles scattered through a rather stiff chocolate clay. A small amount of chert particles is also present. Table 7 gives a screen analysis of the manganese-bearing clay and shows the extreme fineness of the material. Approximately 55 percent is finer than 200-mesh; and this fine material, although lower in manganese than the coarser sizes, contains about 40 percent of the total manganese.

TABLE 7. - Screen Analysis of Manganese-Bearing Clay

Size, mesh	Weight, percent	Assay, percent				Percent of total		
		Mn	Fe	Insol.	P	Mn	Fe	Insol.
Plus 48.....	18.8	10.0	21.6	18.9	---	30.5	34.1	8.3
48 to 100...	9.4	8.0	15.2	33.5	---	12.1	12.0	7.3
100 to 200..	16.6	6.4	11.1	42.5	---	17.2	15.4	16.4
Minus 200...	55.2	4.5	8.3	53.0	---	40.2	38.5	68.0
Composite.	100.0	6.2	11.9	43.0	1.04	100.0	100.0	100.0

An attempt was made to concentrate this ore by screening out the material coarser than 100-mesh and tabling it, but the tabling was not commercially successful. Hydraulic classification as undertaken by the operators was tried, but in no instance could a satisfactory separation be made.

Due to the large percentage of the manganese in the form of extremely fine particles gravity concentration does not appear promising. The flotation process also offers very little hope, as the clay which forms the bulk of the material absorbs large quantities of reagents, tends to come up in the froth, and gives a siliceous concentrate.

Wad ore. - A considerable amount of ferruginous manganese ore has been mined in the past and is now being produced in the district. The large lumps of high-grade ore are generally picked from these ores and shipped separately. The greater part of the material can best be described as wad ore and is characterized by a lower manganese and higher iron, phosphorus, and insoluble content than the lump ores. The material is also quite soft and porous and usually has a moisture content of approximately 30 percent.

An average analysis of a shipment of 1250 tons of wad ore, during 1928 is as follows: Manganese, 25.4 percent; iron, 16.4 percent; silica, 15.5 percent; alumina, 9.44 percent; and phosphorus, 1.18 percent. The moisture content was 29 percent.

Due to the readiness with which the wad ores disintegrate gravity concentration has questionable value. The silica content could undoubtedly be reduced somewhat, but a large portion of the manganese would be lost in the slime. The high moisture content of the wad ores costs the producers an appreciable sum each year in added freight charges. The erection of a small custom-sintering plant in the district would seem justified. The use of nothing more than a drying plant would have no value, as removal of the moisture would render the ore so dusty that it would be difficult to handle, and the dust losses would be great.

Deposits in Cason Shale and Its Residual Clay. - The following description of the deposits in the Cason shale is taken from Miser (260, pp. 57-58), who states:

Manganese and ferruginous manganese ores are widely distributed in the Cason shale. The deposits have been worked at a number of places and have yielded most of the ferruginous manganese ores and a considerable part of the manganese ores that have been produced in the Batesville district. The Cason shale is generally present in the vicinity of Cushman and farther west. East of Cushman it is absent at most places, but the ore deposits in that area are larger and contain a higher manganese content and a lower iron content than the deposits farther west. The shale is thin, at no place exceeding $12\frac{1}{2}$ feet in thickness, but its residual clay is 20 feet or more thick at the Montgomery mine, suggesting that the shale at that locality was probably more than $12\frac{1}{2}$ feet thick.

The ores consist of iron and manganese oxides, which are in places more or less intimately mixed. The manganese oxides are mainly psilomelane and the iron oxides include both red and brown oxides. These minerals occur as irregular masses, as thin horizontal seams and beds, and as "buttons", which are flattened concretion-like masses about an inch in their longest diameter.

A pilot mill was constructed in 1927 by M. R. W. Mining Co. of Memphis to concentrate the "button" ore of the Cason mine. The flow sheet of the mill was very similar to that of a mill in the tri-State lead and zinc district, and consisting of crushers, rolls, rougher jigs, cleaner jigs, and tables. The

capacity of the mill was rated at 40 tons of crude ore per shift. Reports (85) state that during the time the mill operated 15 tons of concentrates, assaying 37 to 40 percent manganese, were recovered from each 100 tons of feed. The mill was operated only a short time and was not a success.

Analyses of the shale and the "buttons" are taken from tables by Miser (260, pp. 139-140) and given in table 8. The percentage of manganese in the "buttons" varies greatly, ranging from 5.85 percent in the red "buttons" to 50.41 in the black. The percentage of manganese in the greenish-gray shale in which the "buttons" occurred was 5.87 percent.

TABLE 8. - Analyses of Shale and "Buttons" from Cason Mine

Material			Assay, percent			
	Mn	Fe	SiO ₂	Al ₂ O ₃	CaO	P
Greenish-gray shale..	5.87	4.98	32.73	8.22	14.22	---
Red "buttons".....	5.85	3.08	2.73	5.68	29.85	---
Black "buttons".....	50.41	7.56	12.67	1.27	2.09	0.06

The Cason mine was inspected in March 1929. The mine was not in operation, and it appeared that much of the ore containing the relatively high-manganese black "buttons" had been removed. The shale exposed in the opencuts was largely greenish gray, and the "buttons" were only partly enriched by manganese. Evidently much of the ore in the Cason shale is of this type, and samples were accordingly taken to determine whether the material could be concentrated.

Concentration tests. - The ore from the unaltered opencut faces is hard, and the "buttons" do not shell out during crushing. Fracturing generally occurs through the "buttons" which contain on the average not more than 25 percent of manganese. A representative sample of the ore contained: manganese, 20 percent; iron, 7.5 percent; insoluble, 32.3 percent; and phosphorus, 0.32 percent.

Float-and-sink tests of hard, unweathered, Cason button-bearing shale showed that high-grade concentrates could not be made by gravity concentration. Ore sized between 35- and 65-mesh gave a heavy liquid (specific gravity, 2.92) concentrate assaying 26.8 percent manganese and 23.7 percent insoluble. When heavier liquids were used the percentage of manganese recovered decreased rapidly, and the grade of concentrate was not much improved.

Replacement Deposits in Fernvale Limestone. - Some manganese occurs in replacements in the Fernvale limestone, but the production has not been large. Where the Fernvale limestone outcrops disseminated manganese oxide can be seen. Samples of such material show that the manganese tenor was less than 5 percent.

The total amount of manganese in the Batesville district is large, but the tonnage of ores available under present mining and milling conditions is stated to be only about 420,000 tons. The amount of manganiiferous clay probably runs into millions of tons, but the low manganese tenor of this material and the absence of any proved metallurgical process for its treatment make this material of doubtful value at present. The development of a low-cost treatment process would make these deposits valuable. Miser (260, p. 85), in quoting from his report of 1918 on the Batesville-Cusiman district, states:

An estimate of the quantity of available manganese ore of all grades in this region where mining is not preceded by systematic prospecting is difficult to make. Of the 180 deposits examined, about half contained an estimated available reserve of 200 tons or less. Only about one third contain 1,000 or more tons and only a few contain more than 5,000 tons, though certain of these contain many thousand tons. A small number of prospects and mines, however, were not visited, and these and the unexplored deposits may increase considerably the reserve. The deposits of the region perhaps include at least 250,000 tons of available ore containing 40 percent or more manganese and 170,000 tons of available ore containing less than 40 percent manganese.

In computing these figures Miser considers the cost of mining and preparing the ore for market and accordingly calculates a smaller tonnage of low-grade than of high-grade ore. Since this estimate was made new ore has been developed, but it is believed that the amount of new ore which can be concentrated does no more than offset the production from this district in the 13 years since Miser's estimate was made.

A fairly steady production of the same magnitude as in 1928 and 1929 probably can continue from the district for years to come if the value of ore remains at the price level prevailing during those years. The possibility of a large increase in production in the near future under normal conditions does not seem great, due largely to the failure of all attempts to put mining of the high-grade ores on an economic large-scale production basis and to the fact that much of the lower-grade ore will not respond to known methods of concentration.

Leadville District, Colo.

Colorado has been one of the larger producers of ores of manganese. Furness (105) estimates that the total production of the State up to and including 1925 has been 28,799 tons of manganese ore containing 35 or more percent manganese, and 3,088,078 tons of ore containing 5 to 35 percent manganese. Since that time Colorado has produced 85,852 tons (375) of ferruginous manganese ore, exclusive of fluxing ore, up to January 1, 1931.

The Leadville area has been the largest shipper of ores of manganese in Colorado. Much of it is of the ferruginous manganese type, and it is well-suited for the manufacture of spiegeleisen. During the war much of it was used to make a spiegeleisen with 33 percent manganese.

The chief producer of ferruginous manganese ore during recent years has been Cramer & Co., which operates the Star group of mines. During 1917 and 1918 this company produced about 250 tons of ore daily. Recent production has been 50 to 100 tons a day.

A representative analysis of ore from this property is: Manganese, 19.6 percent; iron, 20.4 percent; silica, 7.41 percent; phosphorus, 0.043 percent; and moisture, 17 percent. Ore from the Leadville district finds its market at the Pueblo (Colo.) plant of the Colorado Fuel & Iron Co.

Due to the inability to obtain representative samples of ore from this district no experimental work was done to see if the ore could be concentrated to a higher grade.

Cartersville District, Ga.

Location and Extent

For many years it has been known that numerous manganese deposits occur in northeastern Georgia, and the production of ores of manganese from that State is recorded as early as 1865. Of the several districts in this section the Cartersville area, near the town of the same name, has been the principal producer.

The ore deposits of the Cartersville district are found in a belt approximately 18 miles long and 1 to 2 miles wide. The longer axis extends north and northeast, the town of Cartersville lying just west of the district and about 4 miles from the southern boundary.

According to LaForge (172) workable manganese deposits are found in an area of about 30 square miles and occur in the clay and unconsolidated material resulting from the decay of the Weisner quartzite and Shady limestone.

Mineralogy

Various manganese minerals are found, the most common being psilomelane, pyrolusite, manganite, and wad. The greater part of the manganese is believed to occur as psilomelane, although in restricted localities pyrolusite predominates. The physical form in which manganese minerals occur range from soft powdery ore to hard nodular material, the latter predominating. With the manganese minerals in the residual clay varying amounts of cherty material are usually found. Limonite is commonly present and in some deposits is more abundant than the manganese oxides. Barite is present in some deposits, but the barium content of many ores is due to the barium content of the mineral psilomelane.

The largest production of manganese ores and ferruginous manganese ores during recent years has come from mines of the Georgia Manganese & Iron Co., which has been active in modernizing its mining methods and in the erection of a large modern mill, which was completed late in 1929. According to company officials this mill has a rated annual capacity of 200,000 tons of ore.

The ore is mined from open pits by hydraulic methods and washed to a central sump in the bottom of the pit, where coarse ore and rock are removed on a grizzly. Ore fine enough to pass the sump grizzlies is pumped to log washers for removal of clay. The granular material passes to the concentrator, where the coarser portion of the ore is jigged and the fines are tabled. During 1930 the mill produced approximately 12,000 tons of concentrates.

Concentration Tests

Gravity Concentration. - The production of manganese concentrates low enough in silica to meet metallurgical requirements is not unduly difficult. The manganese and gangue minerals are not intimately locked; and fine crushing, except on middling products, is not required. Before the new mill was built some difficulty was experienced in the pilot plant in making satisfactory table concentrates. A study in 1928 of the table feed at Rolla indicated that if the ore were classified before tabling no great difficulty would be experienced. This problem is believed to have been solved satisfactorily in the new mill.

Magnetic Concentration. - Limonite is associated with the manganese oxides. It is particularly abundant in the upper portions of the deposits, and care is taken not to mine these iron zones. Some iron, however, is always associated with the ores and usually contaminates the mill concentrate. Limonite and other iron-bearing minerals are difficult to remove by jigs and tables. Inasmuch as ore having a ratio of less than 9 : 1 (preferably 10 : 1) parts of manganese to iron cannot be used for the manufacture of standard-grade--80 percent--ferromanganese considerable attention was given to a study of ore containing a greater ratio of iron to manganese than that mentioned. A brief general report on this phase of the manganese investigation was published in 1929 (68).

Although iron in manganese ores or concentrates is not penalized directly, the manganese in the ore is paid for on a sliding basis, with the result that an indirect penalty is in effect. For example, an ore containing 30 percent manganese and 20 percent iron may be paid for at a rate of 40 cents per unit of manganese, no credit being given for the iron content, whereas an ore containing 48 percent manganese and 5 percent iron will be worth 55 cents per unit. Therefore, the producer gains by shipping as high-grade a concentrate as possible.

The specific gravity of the limonitic grains in the Cartersville ores is 3.4 to 3.85, while that of the manganese particles is between 3.7 and 4.4. With such a small difference in specific gravity a clean separation between the two minerals cannot be made on jigs and tables. A roasting and magnetic process, therefore, seemed the only logical way to make the separation, and tests were accordingly made on the jig concentrates.

Six samples of jig concentrates were obtained from the pilot plant of the Georgia Manganese & Iron Co. in 1928. Each sample, which represented jig concentrates made at various times and on different types of ore, was given a

reducing roast at 500°C. for 30 minutes. The roast converted the limonite into magnetic oxide (magnetite). After roasting the ore was quenched and separated for a magnetic and nonmagnetic product. No crushing was done before or after roasting. The results of these tests, given in table 9, show that the jig concentrates are particularly adapted to roasting and magnetic separation. From each sample a magnetic iron product and a nonmagnetic concentrate high in manganese and low in iron and well-suited for the manufacture of 80 percent ferromanganese were made. Fortunately the iron minerals are associated more closely with the phosphorus-bearing and siliceous minerals that are the manganese oxides. Thus, the manganese concentrate is lower than the roasted ore in both objectionable elements. The magnetic iron concentrate may be marketed as ferruginous manganese ore.

Some jig concentrates would not have to be submitted to a roasting and magnetic process. Much of the ore--for example, the first three jig concentrates in table 9--is so close to the required ferrograde that it could be mixed with the treated concentrates from more ferruginous ores, and the combined concentrates would still meet the requirements.

Magnetic separation would not supplant the jigging now in vogue, because the silica must first be eliminated. Free siliceous gangue particles in a gravity concentrate would pass into the nonmagnetic manganese product. The method does, however, make possible the production of a high-manganese, low-iron concentrate and would allow the operators to work ore bodies relatively high in iron, now neglected or wasted.

In brief, the Cartersville district of Georgia contains large bodies of ferruginous manganese ore in residual deposits that have as yet been only partly developed. A mining method whereby the ore can be removed from open pits by hydraulic methods has been successfully worked out. A modern mill was erected during 1929 and in 1930 produced about 12,000 tons of concentrates. The concentrates are suitable for many metallurgical uses, and laboratory work shows that the ferruginous concentrates sometimes produced can be converted to a high-grade ferromanganese ore and a manganiferous iron ore by roasting and magnetic separation.

Menominee District, Mich.

Location and Extent

The Menominee district, Mich., west of Escanaba, is primarily an iron-ore producer. However, in this area, as in other Lake Superior districts, manganiferous iron ores are present. The district also contains ferruginous manganese ores which, on account of their relatively high manganese content, were given attention.

Concentration Tests

These ores, which are high in manganese, are quite low in silica but high in phosphorus, therefore the purpose of the investigation was to determine if a high-grade manganese concentrate could be recovered low enough in phosphorus to be suitable for the manufacture of ferromanganese.

TABLE 9. - Roasting and Magnetic Test of Manzanese Jig Concentrates

Approximate size, inch	Product	Weight, percent	Assay, percent			Percent of total			
			Mn	Fe	Insol.	P	Mn	Fe	Insol.
3/4.....	Unroasted ore.....	---	46.20	5.0	4.1	0.142	---	---	---
	Magnetic concentrate...	7.8	29.3	27.9	8.6	.268	4.5	47.6	13.7
	Nonmagnetic concentrate	92.2	52.9	2.6	4.6	.136	95.5	52.4	86.3
	Composite.....	100.0	51.1	44.6	4.9	0.146	100.0	100.0	100.0
3/8.....	Unroasted ore.....	---	44.8	5.3	4.5	.141	---	---	---
	Magnetic concentrate...	15.0	28.6	27.0	9.0	.227	5.5	74.9	27.4
	Nonmagnetic concentrate	85.0	52.2	1.6	4.2	.128	94.5	25.1	72.6
	Composite.....	100.0	48.7	5.4	5.0	0.143	100.0	100.0	100.0
3/16.....	Unroasted ore.....	---	44.8	6.3	3.7	.141	---	---	---
	Magnetic concentrate...	5.1	14.4	44.9	4.3	.210	2.0	37.1	5.9
	Nonmagnetic concentrate	94.9	51.2	4.1	3.7	.140	98.0	62.9	94.1
	Composite.....	100.0	49.3	6.2	3.8	0.144	100.0	100.0	100.0
1.....	Unroasted ore.....	---	32.8	17.9	7.4	.170	---	---	---
	Magnetic concentrate...	38.7	12.4	45.5	12.0	.259	13.2	92.6	60.7
	Nonmagnetic concentrate	61.3	51.7	2.3	5.0	.118	86.6	7.4	39.3
	Composite.....	100.0	36.5	19.0	7.8	0.173	100.0	100.0	100.0
1/2.....	Unroasted ore.....	---	31.2	16.8	10.2	.162	---	---	---
	Magnetic concentrate...	30.9	4.0	55.2	12.7	.226	2.7	70.5	35.0
	Nonmagnetic concentrate	69.1	46.9	2.5	10.5	.119	37.3	9.5	65.0
	Composite.....	100.0	33.3	18.9	11.2	0.152	100.0	100.0	100.0
1/4.....	Unroasted ore.....	---	38.5	11.4	4.8	.130	---	---	---
	Magnetic concentrate...	20.1	11.8	47.7	8.8	.199	5.5	82.8	31.7
	Nonmagnetic concentrate	79.9	51.0	2.5	4.8	.116	94.5	17.2	68.3
	Composite.....	100.0	43.1	11.6	5.6	0.133	100.0	100.0	100.0

A ferruginous manganese ore of the type described was obtained from the Judson mine of the Balkan Mining Co., Alpha, Mich. The analysis of the ore shipped by this company during 1929 follows (225): Iron, 35.29 percent; phosphorus, 0.745 percent; silica, 5.47 percent; manganese, 19.25 percent; alumina, 2.57 percent; calcium oxide, 2.33 percent; magnesia, 4.21 percent; sulphur, 0.075 percent; and loss on ignition, 4.62 percent.

The sample examined was approximately of the composition stated above. Visual examination showed that the manganese and iron minerals were locked and that crushing to at least 10-mesh would be necessary for any liberation. The manganese was present in oxide form, much of it being manganite. The iron was present as hematite. A screen analysis of the ore showed that the manganese minerals were harder than the hematite and tend to segregate in the coarser sizes.

Gravity concentration. - Float-and-sink tests were made of sized portions of the ore crushed to 10-mesh to determine whether phosphorus could be removed by a gravity process. A liquid with a density of 3.30 was selected so that if the phosphorus occurred as clean apatite (specific gravity, 3.15) it would be removed in the float. The examination indicated that gravity concentration would eliminate some of the phosphorus but that the greater proportion would remain locked with the iron and manganese. The degree of concentration of iron and manganese and the elimination of insoluble and phosphorus are shown in table 10.

Magnetic concentration. - A series of roasting and magnetic concentration tests was made to determine if the iron could be separated from the manganese minerals. A sample crushed to 10-mesh was given a reducing roast at 550°C. for 30 minutes. Tests with the Davis magnetic tube were made of a sample that had not been crushed after roasting, of a sample through 35-mesh, and of a sample through 100-mesh.

The results of these tests are given in table 11 and show how intimately the minerals are associated.

Upon crushing to 100-mesh, 61.1 percent of the manganese is still locked with the iron. The nonmagnetic product is high in manganese; the minus 100-mesh product has a tenor of 48.58 percent manganese. The phosphorus and insoluble tend to be segregated to about the same degree in the nonmagnetic portion.

To obtain more information on the occurrence of phosphorus two sized lots, 28- to 65-mesh and 65- to 150-mesh, were given a magnetic roast similar to that previously mentioned. Magnetic tube tests were then made of each lot. The nonmagnetic tailing from each size was then fractionated with a heavy liquid with a density of 3.30. The results are shown in table 12.

TABLE 10. - Float-and-Sink Tests of Judson Ore

Size, mesh	Product	Weight percent	Assay, percent			Percent of total		
			Mn	Fe	Insol.	P	Mn	Fe
10 to 28.....	Float on 3.30.....	5.4	7.82	20.53	17.20	1.610	1.7	3.4
	Sink in 3.30.....	94.6	25.37	33.73	3.32	.482	98.3	96.6
	Total.....	100.0	24.42	33.02	4.07	0.543	100.0	100.0
48 to 100.....	Float on 3.30.....	7.4	7.70	18.85	24.52	1.720	2.7	4.0
	Sink in 3.30.....	92.6	22.13	36.21	3.80	.485	97.3	96.0
	Total.....	100.0	21.06	34.92	5.33	0.576	100.0	100.0

TABLE 11. - Magnetic Tube Tests of Roasted Judson Ore

Size, mesh	Product	Weight percent	Assay, percent			Percent of total		
			Mn	Fe	Insol.	P	Mn	Fe
Minus 10.....	Magnetic concentrate....	87.0	19.00	42.36	4.64	0.561	74.4	96.4
	Tailing.....	13.0	43.80	10.61	7.60	1.029	25.5	3.6
	Total.....	100.0	22.22	38.23	5.03	0.622	100.0	100.0
Minus 35.....	Magnetic concentrate....	84.7	17.87	42.56	5.40	.571	68.7	96.3
	Tailing.....	15.3	45.05	8.92	8.40	.973	31.3	3.7
	Total.....	100.0	22.02	37.42	5.85	0.632	100.0	100.0
Minus 100.....	Magnetic concentrate....	82.3	16.41	45.04	4.40	.568	61.1	97.0
	Tailing.....	17.7	48.58	6.45	7.10	1.032	38.9	3.0
	Total.....	100.0	22.11	38.21	4.88	0.650	100.0	100.0

TABLE 12. - Roasting and Magnetic Tests Followed by Float and Sink Tests of Tube Tailings

Size, mesh	Product	Weight, percent	Assay, percent				Percent of total			
			Mn	Fe	Insol.	P	Mn	Fe	Insol.	P
28 to 65	(Magnetic concentrate	86.0	18.93	41.86	4.80	0.57	69.6	97.1	79.7	85.2
	(Tube tailing	14.0	50.77	7.60	7.50	.606	30.4	2.9	20.3	14.8
	(Total .	100.0	23.39	37.06	5.18	0.576	100.0	100.0	100.0	100.0
	(
	(
28 to 65	(Tube tailing:									
	(Float on 3.30	7.8	7.80	15.38	39.30	1.704	1.2	15.8	40.9	22.0
	(Sink in 3.30	92.2	54.41	6.94	4.80	.513	98.8	84.2	59.1	78.0
	(Total .	100.0	50.77	7.60	7.50	0.606	100.0	100.0	100.0	100.0
	(
65 to 150	(Magnetic concentrate	87.2	19.59	42.66	4.60	.582	70.8	97.9	80.2	86.7
	(Tube tailing	12.8	52.12	6.10	7.73	.611	29.2	2.1	19.8	13.3
	(Total .	100.0	22.83	37.99	5.00	0.586	100.0	100.0	100.0	100.0
	(
	(
65 to 150	(Tube tailing:									
	(Float on 3.30	9.8	7.20	16.63	38.06	2.210	1.4	26.8	53.3	35.5
	(Sink in 3.30	90.2	57.00	4.96	4.00	.437	98.6	73.2	46.7	64.5
	(Total .	100.0	52.12	6.10	7.73	0.611	100.0	100.0	100.0	100.0
	(

In the tests of 65- to 150-mesh material it was technically possible to produce a magnetic manganiiferous iron concentrate assaying 42.66 percent iron, 18.59 percent manganese, 4.60 percent insoluble, and 0.582 percent phosphorus and also to produce a manganese concentrate with a content of 4.96 percent iron, 57.00 percent manganese, 4.00 percent insoluble, and 0.437 percent phosphorus. Unfortunately the high phosphorus content renders this concentrate useless for the manufacture of ferromanganese.

Cuyuna District, Minn.

Location and Extent

Its large reserves and production make the Cuyuna district, Minn., one of the more important manganese districts of the country. In 1928 and 1929 the production of manganiiferous iron ores was approximately 1,000,000 tons annually. The annual production of ferruginous manganese ores during this same period was about 5,000 tons.

The district lies southwest of the Mesabi range and extends southwest 65 miles from Aitken through Crosby and Brainerd. The district is 1 to 12 miles wide. The area is of comparatively recent discovery, having been found by drilling areas that had been mapped by magnetic surveys. The surface of the district is rolling, and all ore bodies are covered by a heavy mantle of glacial drift. The occurrence of the manganese-bearing ore has been described by Zapffe (403).

Types of Ore

The manganese-bearing ores of the Cuyuna district may be grouped roughly into two classes. The most abundant ore is the low-silica, low-manganese, high-phosphorus type called brown ore. The other type is the so-called black ore, which is characterized by a higher manganese and silica content and by a lower phosphorus analysis than the brown ore. As a general rule the brown ore has less than 10 percent, but sometimes as high as 22 percent manganese is present.

Although many of the black ores have a relatively high iron content they are not merchantable on account of their high silica content. Ores of this type are frequently found in existing mines, and their concentration presents a pressing problem, because, should these mine workings be allowed to cave before this ore is removed, the cost of its recovery will be greatly increased. The concentration studies have accordingly been limited to the black ores.

Much of the data in the following pages has been taken from a previous report (67) published in 1930.

Mineralogy

The black ores consist of intimately associated iron, manganese, and gangue minerals. Hematite and psilomelane are the predominating ore minerals. Small quantities of manganite, pyrolusite, and limonite are also present. The

hematite is usually of micaceous habit and occurs disseminated in widely varying proportions in quartz, carbonate minerals, and psilomelane. Psilomelane occurs disseminated in a manner similar to hematite, except that the psilomelane is usually in coarser grains.

The more important gangue minerals are quartz, feldspar, and mixed carbonates, quartz predominating. Feldspar occurs principally in large grains, sometimes partly replaced by psilomelane. The carbonates are often found as cementing material for fine quartz grains and for hematite, psilomelane, manganite, and pyrolusite particles. Calcite is the predominating carbonate, with minor amounts of rhodochrosite and siderite.

Much of the gangue may be characterized as a ferruginous chert. This material usually contains enough iron oxide to give it rather high specific gravity, making removal by gravity methods difficult. A further difficulty in ore testing is the similarity of the mineral and gangue grains, making visual examination almost futile; many chemical analyses are required.

Some idea of the close association of the manganese and iron minerals may be given by stating that on crushing to 325-mesh 40 percent of the manganese still remains locked with the iron. Upon crushing the manganese tends to concentrate in the coarser sizes, due to the hardness of the grains of high manganese content.

Concentration Tests

In a study of the characteristics of the ore to determine the degree of locking of the mineral and the grade of concentrate which could be made, various methods were employed. Microscopic examinations of polished sections were made. Roasting and magnetic concentration tests were performed, and detailed dissections of the ore were made by the use of heavy liquids.

Microscopic Examination. - Microscopic examinations were made of polished sections and the results recorded by photomicrographs and camera lucida drawings. Photomicrographs and camera lucida drawings are shown in figures 5, A and B, and 6, A and B, of briquets from two Cuyuna range ores. These figures show the extreme locking which exists between the mineral constituents of the ore. The results of this microscopic study have been given in detail in an earlier paper by Cooke and associates (58).

Float-and-Sink Tests. - Float-and-sink tests with various heavy liquids brought out many interesting facts. In general, they showed that the black ores have an appreciable amount of relatively clean mineral and also some clean gangue. Some of the gangue, however, is a ferruginous chert having a specific gravity not much less than that of manganese and iron oxides. The rejection of such heavy gangue material by gravity methods would be difficult. Float-and-sink tests of a representative ore are shown in table 13. This ore was crushed to pass a 4-mesh screen, and heavy liquid separations were made on sized samples. Material finer than 200-mesh was not treated. The results in table 13 represent the combined specific gravity fractions from 4- to 200-mesh. Material having a specific gravity greater than 4.0 assays 24.54 percent

manganese, 34.98 percent iron, and 6.51 percent silica. This fraction contains 70.2 percent of the manganese and 51.0 percent of the iron. The presence of much locked material is indicated by the relatively large fraction having a specific gravity between 3.65 and 4.0 and it is made up partly of locked particles and to a greater extent of ferruginous chert. The material lighter than 3.0 consists mainly of clean gangue, the removal of which would be relatively easy with any gravity process.

Float-and-sink tests offer an excellent method of determining the association of gangue and the heavier manganese and iron minerals. The specific gravities of the iron and manganese minerals are, however, so nearly the same as to make this method impractical for determining the degree of grinding necessary to obtain a liberation between the iron and manganese minerals. This information was therefore found by roasting the ore to convert the iron oxides to magnetite, which was later removed magnetically.

Magnetic Concentration. - A sample was given a magnetic roast and then divided into six portions. Each portion was crushed to pass a certain screen, after which a magnetic separation was made. These tests, listed in table 14, show the extremely fine interlocking of the manganese and iron minerals. The degree of liberation progresses continuously as the size of ore particles decreases. Material crushed to pass a 14-mesh screen showed that only 15.83 percent of the manganese was liberated from the magnetic iron concentrate, whereas the sample crushed to pass 325-mesh showed liberation of 61.65 percent of the manganese.

The percentage of gangue in the magnetic concentrate also decreases in the finer sizes. The 14-mesh concentrate had 12.36 percent of insoluble, whereas the concentrate from the 325-mesh size contained 7.20 percent of insoluble. At 14-mesh, 48.61 percent of the insoluble is liberated from the iron concentrate. This amount increases progressively until at 325-mesh, 78.47 percent has been unlocked.

The manganese and iron minerals are associated so closely as to preclude the production of a concentrate of ferro grade by ore-dressing methods. It should be possible to make manganese concentrates considerably enriched in respect to the iron, but production of a concentrate containing 9 or 10 times as much manganese as iron, such as is required for a ferromanganese ore, seems impossible. The association of the silica with the valuable minerals in most of the ores examined is not as intimate as the manganese-iron combination, hence the production of concentrates low enough in silica for some metallurgical purposes seems possible.

Concentration by Classification and Tabling. - Without resorting to classification, good tabling results may be obtained on clean sulphide ores that have large differences between the specific gravities of the gangue and the mineral. On the other hand, due to the presence of heavy gangue particles and much locked mineral the Cuyuna ores present a difficult problem. For good concentration the ore must be well-classified before tabling is undertaken.

TABLE 13. - Float-and-Sink Tests of a Cuyuna Ore, 4 to 200-Mesh

Specific-gravity fraction	Weight, percent	Assay, percent			Percent of total		
		Mn	Fe	SiO ₂	Mn	Fe	SiO ₂
Float on 3.00	21.12	0.46	5.33	90.81	0.6	4.1	57.1
3.00 to 3.65	13.41	5.22	30.37	44.33	4.9	14.6	17.7
3.65 to 4.00	13.30	19.67	31.02	18.86	18.4	14.8	7.5
Sink in 4.00	40.67	24.54	34.98	6.51	70.2	51.0	7.9
Composite	88.50	15.12	26.60	34.21	94.1	84.5	90.2
Slime (minus 200-mesh)...	11.50	7.32	37.89	28.63	5.9	15.5	9.8
Total	100.00	14.23	27.90	33.57	100.0	100.0	100.0

TABLE 14. - Magnetizing Roast and Magnetic Separation Test of Minus 10-Mesh Ore Roasted at 550° C. for 30 Minutes

Size, mesh	Product	Weight, percent	Assay, percent			Percent of total			
			Mn	Fe	Insol.	SiO ₂	Mn	Fe	Insol.
All through: 14	Nonmagnetic tailing	18.3	14.26	6.86	52.20	- -	15.83	3.72	48.61
	Magnetic concentrate	81.7	16.98	39.69	12.36	- -	84.17	96.28	51.39
	Composite	100.0	16.48	33.68	19.65	- -	100.00	100.00	100.00
28	Nonmagnetic tailing	19.5	16.88	5.68	50.60	- -	19.71	3.31	50.91
	Magnetic concentrate	80.5	16.66	40.18	11.82	- -	80.29	96.69	49.09
	Composite	100.0	16.70	33.45	19.38	- -	100.00	100.00	100.00
48	Nonmagnetic tailing	22.0	18.06	5.78	48.40	- -	24.18	3.74	55.20
	Magnetic concentrate	78.0	15.97	41.94	11.08	- -	85.82	96.26	44.80
	Composite	100.0	16.43	33.98	19.29	- -	100.00	100.00	100.00
100	Nonmagnetic tailing	26.6	21.92	4.90	42.50	- -	35.31	3.82	59.56
	Magnetic concentrate	73.4	14.55	44.69	10.46	- -	64.69	96.18	40.44
	Composite	100.0	16.51	34.11	18.98	- -	100.00	100.00	100.00
200	Nonmagnetic tailing	36.0	23.45	5.29	38.80	- -	51.63	5.70	71.63
	Magnetic concentrate	64.0	12.36	49.20	8.64	- -	48.37	94.30	28.37
	Composite	100.0	16.35	33.39	19.50	- -	100.00	100.00	100.00
325	Nonmagnetic tailing	40.8	24.39	5.78	38.10	- -	61.65	7.03	78.47
	Magnetic concentrate	59.2	10.46	52.63	7.20	- -	38.35	92.97	21.53
	Composite	100.0	16.14	33.52	19.81	- -	100.00	100.00	100.00
	Unroasted ore	- -	15.04	32.14	20.76	18.62	- -	- -	- -

TABLE 15. - Table Concentration of 10-Mesh Classified Feed

Product	Weight, percent	Assay, percent			Percent of total		
		Mn	Fe	SiO ₂	Mn	Fe	SiO ₂
Table concen- trates	54.79	20.87	34.62	9.74	72.07	59.98	27.20
Table tailings ...	24.29	10.92	23.86	33.73	16.72	18.32	41.78
Table feed	79.08	17.81	31.32	17.10	88.79	78.30	68.98
Slimes	20.92	8.50	32.80	29.08	11.21	21.70	31.02
Composite	100.00	15.86	31.63	19.61	100.00	100.00	100.00

Tabling certain of these ores has made a merchantable grade of concentrates with a fair recovery. Due to the necessity of fine grinding to obtain liberation, some of the ores, however, are not amenable to such treatment. The results of tabling one of the more amenable ores are shown in table 15.

The ore was crushed to 10-mesh and classified into four spigot products and a slime overflow. Each spigot product was tabled separately and a series of samples taken around the table. These cuts were later combined according to analysis, and for simplicity only the composite concentrates and tailings are shown. The combined concentrates represented 54.79 percent of the weight of the samples and had a tenor of 20.87 percent manganese, 54.62 percent iron, and 9.74 percent silica. Approximately 72.0 percent of the manganese and 60 percent of the iron were recovered. In this test a small laboratory table was used, and none of the middlings were re-treated. As already mentioned, a better recovery of manganese than iron is shown. A commercial plant would have the advantage of classifying the ore into a greater number of products of closer size, of using larger tables, and of grinding and recirculating middlings.

Upon crushing an ore to table size slimes are unavoidably produced, and unless some auxiliary method is available for the treatment of these slimes much mineral is lost. On some ores examined locking of the mineral was so intimate that grinding to at least 100-mesh would be necessary to give a concentrate of commercial grade. Flotation was accordingly tried on table slimes and on ores ground directly to flotation size.

Flotation. - A flotation test of a high-manganese nonmerchantable ore showed that a desirable concentrate could be made. The results are given in table 16. The concentrate contained 67.84 percent of the manganese and 53.11 percent of the iron. The middlings were not recleaned.

TABLE 16. - Flotation Test of a High-Manganese Ore

Product	Weight, percent	Assay percent			Percent of total		
		Mn	Fe	SiO ₂	Mn	Fe	SiO ₂
Cleaner concentrates...	53.4	29.16	25.53	9.77	67.84	53.11	26.87
Middlings.....	17.9	22.51	27.35	19.75	17.56	10.06	18.22
Rougher concentrates...	71.3	27.49	25.99	12.28	85.40	72.17	45.09
Rougher tailings.....	28.7	11.68	24.90	37.20	14.60	27.83	54.91
Composite.....	100.0	22.95	25.63	19.43	100.00	100.00	100.00

<u>Reagents used</u>	<u>Pounds per ton of ore</u>
Oleic acid.....	0.18
Sodium oleate.....	.50
Pine oil.....	.24
Sodium silicate.....	3.00
Sodium carbonate.....	2.00

The reagents used evidently had little tendency to float the iron minerals since the iron analyses of the concentrates and tailings were nearly the same. It is thought that much of the iron that went into the concentrates did so solely because it was locked with manganese minerals; that is, the flotation is selective with respect to the manganese. The tendency of the manganese minerals to float is pronounced. The concentrates had 29.16 percent manganese, while analysis of the tailings showed only 11.60 percent.

Flotation requires the use of soft water. If hard water containing large quantities of calcium or magnesium salts is used the reagent consumption is increased due to precipitation of the oleates as insoluble calcium or magnesium oleates, and the time required for the frothing operation is also increased. With soft or treated waters the manganese minerals float fairly rapidly. Then the amount of reagents used decreases, and a more efficient separation can be made. Table 17 shows the results of a test of a Cuyuna ore, in which softened water was used.

Flotation offers a means of concentrating some ores which are so badly locked as to prevent a merchantable concentrate being made without fine grinding. Table 18 shows the results of float-and-sink tests of an ore of this nature sized between 35- and 65-mesh. The high silica content of the two heavier fractions indicates that there is little hope of obtaining a concentrate from this ore by tabling at 10-mesh. Actual tabling tests proved this point; the concentrates carried about 15 percent silica. Grinding to 100-mesh, however, liberated the mineral enough to allow a merchantable concentrate to be made by flotation. Flotation cannot, however, be used with success on all Cuyuna ores. A few of the ores examined are so complex that grinding to 100-mesh does not make possible the production of concentrates low enough in silica to be merchantable at present. One ore, which is representative of rather large reserves, responded to a tabling treatment but would not yield good-grade concentrates when the entire sample was ground to a fine size and floated. A microscopic study of this ore by Cooke previously mentioned (58) showed that the reason for this was the presence of a small amount of carbonate. This carbonate was very finely disseminated through siliceous gangue particles (see fig. 5, B), and since the reagents used in flotation tended to float carbonate minerals this siliceous carbonate was concentrated in the froth along with the manganese and iron minerals; hence, the resulting concentrates were too siliceous to be acceptable. An all-flotation process therefore could not be used on this ore, at least not with the reagents now in use. This ore can, however, be tabled first and the objectionable carbonate removed, after which the concentrates can be floated satisfactorily, as explained more fully in the following pages.

Combined Tabling and Flotation. - Some Cuyuna ores contain an appreciable amount of manganese and iron minerals recoverable at table size. It is logical to recover as much as possible of this coarse mineral without further grinding. Another point in favor of using tables instead of crushing direct to flotation size is that the flotation reagents so far used show little tendency to float the iron minerals, whereas some may be recovered by gravity. Consequently, tabling before flotation should materially increase the percentages of iron recovered.

TABLE 17. - Flotation Test, Using Treated Water

Product	Weight, percent	Assay, percent			Percent of total		
		Mn	Fe	SiO ₂	Mn	Fe	SiO ₂
Concentrates.....	73.4	21.80	37.07	8.20	88.69	80.08	33.08
Tailings.....	26.6	7.65	25.45	45.80	11.31	19.92	66.92
Composite.....	100.0	18.04	33.98	18.20	100.00	100.00	100.00

<u>Reagents used</u>	<u>Pounds per ton of ore</u>
Oleic acid.....	0.16
Sodium oleate.....	.50
Pine oil.....	.08
Sodium carbonate....	3.00

TABLE 18. - Float-and-Sink Tests of a Cuyuna Ore, 35- to 65-Mesh

Specific-gravity fraction	Weight, percent	Assay, percent			Percent of total		
		Mn	Fe	Insol.	Mn	Fe	Insol.
Float on 2.95.....	7.2	1.56	10.36	79.40	0.6	2.5	23.6
2.95 to 3.25.....	8.5	4.02	20.92	61.40	1.8	6.0	21.5
3.25 to 3.66.....	15.3	11.80	26.59	40.20	9.5	13.7	25.3
Sink in 3.66.....	69.0	24.30	33.37	10.40	77.8	88.1	29.6
Total.....	100.0	19.06	29.58	24.27	100.0	100.0	100.0

The ore used to determine the value of tabling before flotation was one of the most difficult of the Cuyuna ores examined. Previous larger-scale tests, in which the ore had been crushed to 8-mesh, classified into 10 spigot products, and then tabled, showed that the best concentrates obtainable at this size contained 14.6 percent of silica, clearly indicating the necessity of finer grinding. A sample of the ore was therefore ground to 65-mesh, then deslimed. The granular portion was tabled for concentrates, middlings, and tailings. The middlings were then reground and added to the slimes, which were then treated by flotation. The results of this test appear in table 19.

The table concentrates were somewhat higher-grade than the flotation concentrates, due partly to the cutting out on the table of the cleanest mineral, permitting all locked grains to pass into the middlings for regrinding and re-treatment in the flotation cells.

Pilot Plant

As a result of this investigation a pilot plant consisting of 10 tables was built near Crosby, Minn., in the summer of 1930 to treat black ferruginous manganese ores. Results with this table plant have in general checked laboratory results.

TABLE 19. - Combined Tabling and Flotation Test

Product	Weight, percent	Assay, percent			Percent of total		
		Mn	Fe	SiO ₂	Mn	Fe	SiO ₂
Table concentrates.....	28.8	23.29	36.01	8.84	37.43	33.51	11.18
Flotation concentrates..	31.3	22.72	33.60	12.57	39.68	33.98	17.30
Combined concentrates...	60.1	23.00	34.75	10.75	77.11	67.49	28.48
Table tailings.....	6.2	7.90	17.06	58.58	2.73	3.42	15.98
Flotation tailings.....	33.7	10.71	26.71	37.46	20.16	29.09	55.54
Combined tailings.....	39.9	10.27	25.21	40.73	22.89	32.51	71.52
Composite.....	100.0	17.92	30.95	22.72	100.00	100.00	100.00

<u>Reagents used in flotation test</u>	<u>Pounds per ton of ore</u>
--	----------------------------------

Oleic acid.....	0.20
Pine oil.....	.27
Sodium silicate....	6.00
Sodium carbonate...	5.00

A small flotation unit was also installed to treat the slimes from the table unit. The operation of the flotation unit was not entirely satisfactory, and more large-scale experimental work on this phase of the problem is needed.

This plant produced approximately 25,000 tons of concentrates during the summer of 1931.

Butte District, Mont.

Location and Extent

The Butte district, Mont., first attracted attention as a large potential producer of manganese during 1916 and 1917, when the price of manganese ore was high and the possibilities of obtaining large quantities of domestic ores were first considered. At a comparatively early date some oxidized ores were mined for their silver content, but the use of the black oxidized ore and the pink rhodochrosite ores was not considered seriously as a source of manganese until the beginning of the World War.

The production from this district in 1918 was 72,381 tons, of which the rhodochrosite contributed by far the greater amount. After the war period the mining and shipment of the rhodochrosite continued, but on a smaller scale. The development in 1927 of a process for calcining and nodulizing high-grade rhodochrosite ores constituted a distinct step in the metallurgical process of the district. The development in 1928 of flotation for concentrating the low-grade ores increased the reserves and has made possible production of the highest-grade metallurgical manganese ore in the world market.

The location of the Butte district is well-known to all mining men. Its general geology has been most fully discussed by Weed (381) and Sales (333). A study of the manganese deposits in particular has been made by Pardee (290). In discussing the occurrence and distribution of manganese Pardee (288, pp. 141-146, 174-176) says in part in his later report:

***. The lodes *** are formed along steeply pitching fractures in granite (quartz monzonite) and extend to great depths. Manganese minerals are widely distributed, both horizontally and vertically, but they are comparatively scarce in that part of the district known as the central copper zone and plentiful in an adjacent peripheral belt a mile or more wide called the silver-zinc zone.

Types of Ore

In general, two main types of manganese-bearing ores are found in the Butte lodes, the black oxidized ores near the surface and the pink carbonate and silicate ores that occur below. Pardee (293) says that the lower limit of the zone of oxidation does not extend more than 20 to 200 feet below the surface.

Oxide Ores. - Originally much fairly high-grade oxide ore was found in the outcrops but was largely mined in the early days. Lower-grade ore bodies containing 12 to 35 percent manganese occur, but they have never been opened to any extent as the ore has usually been too low-grade to be marketable, and concentration has not been successful. According to Pardee (288, pp. 173-174), the Butte district was estimated to contain, in 1917, 1,000 tons of high-grade ore (40 percent or more manganese) and about 400,000 tons of siliceous material. The high-grade ore was probably mined during 1918, but only a small part of the low-grade ore was mined.

The character of the oxide ores found in various parts of the district varies considerably, but in general the ores may be said to consist of particles of vein quartz cemented with or partly replaced by a variety of manganese oxides, pyrolusite predominating. The manganese minerals and the siliceous gangue are intimately mixed, making the production of a low-silica concentrate by ore-dressing methods extremely difficult.

Carbonate Ores. - Below the oxidized zone of the lodes in the manganiferous area of Butte the manganese occurs as rhodochrosite, with smaller amounts of manganese silicate (rhodonite). In places the rhodochrosite occurs almost free from impurities. From these ore bodies the high-grade ores--containing 35 percent or more of manganese--were mined during the war period and later were mined and calcined. The deposits of high-grade rhodochrosite ores have been found only in mines in the Black Chief vein. The Emma mine has been the chief producer, although the Ancient and Travona mines produced during 1917-1919. The total amount of high-grade rhodochrosite mined from 1917 through 1928 has been estimated at 230,000 tons.

In addition to the high-grade rhodochrosite deposits larger deposits of ore occur diluted with quartz, some rhodonite, and small amounts of galena, sphalerite, and pyrite. They also carry some silver. The proportions of rhodochrosite, rhodonite, and quartz vary throughout the various mines, and it is impossible to estimate with accuracy the manganese tenor of these deposits.

The occurrence of the low-grade ore is widespread; the amount is indicated by Pardee (288, p. 176), who, in 1921, after discussing the amount of high-grade carbonate ore in the district, said:

*****. In addition the lodes contain an almost unlimited quantity of low-grade material consisting of the carbonate and silicate of manganese and quartz mixed in different proportions. This constitutes a reserve from which, if the necessity arose, the country's needs might be largely supplied. With improvements in the methods of reduction, much of it might even become valuable under ordinary conditions.

On account of the large amount, this low-grade ore has been given the most attention in the recent study of the Butte ores. The results of the experimental work on the oxides and carbonates follow.

Concentration Tests

Oxide Ores. - Much of the oxide ore is of low grade and not usable without concentration. Samples representative of much of the low-grade ore were obtained from outcrops of the Black Chief vein on the Travona claim. The lode was 100 to 150 feet wide. The samples represented 40 to 60 feet of the width. The ore is similar to that of the same vein which has been mined on the Ancient claim and may be deemed a counterpart of the material above the rhodochrosite ores of the Emma mine, where the oxidized zone is not observable.

The composite sample had 12.3 percent manganese, 2.4 percent iron, 75 percent insoluble, and 0.047 percent phosphorus. Much of the manganese occurred as pyrolusite, and the principal gangue mineral was quartz. Microscopic examination showed close interlocking of the manganese minerals and quartz and gave little hope that a low-silica concentrate could be obtained by ore-dressing methods.

Float-and-sink tests of a fraction sized between 20- and 35-mesh substantiated the results of the microscopic examination. The sink in a liquid with a specific gravity of 2.95 contained over 30 percent insoluble, largely silica.

The percentage of manganese in the average Butte oxide ore is low, and the association between the manganese and the quartz is such as to make it extremely difficult, if not impossible, to produce low-silica concentrates by ore-dressing methods. Some oxide deposits are known to be higher-grade and the ore somewhat cleaner, but the small tonnage of this material did not seem to justify further experimental work.

Carbonate Ores. - The low-grade carbonate ores occur as a mixture of rhodochrosite, quartz, and smaller amounts of rhodonite and sulphides. They have no value without concentration and accordingly were thoroughly examined.

Due to the size of the deposit of siliceous carbonate ore in the Emma mine, most of the research work was on ore from that property. A sample of ore requiring concentration was about half quartz; lead, zinc, and iron sulphides were also present. The rhodochrosite amounted to nearly 40 percent, so that the sample had about 20 percent manganese. The sample was taken across a 45-foot vein, no portion of which could be mined to produce a high-grade ore of 36 percent or more manganese content.

Some rhodochrosite occurred as clean pieces 1 inch or larger. Much of the quartz also was present in fairly large pieces. However, some of the quartz was finely disseminated through the rhodochrosite. Galena, sphalerite, and pyrite were found in small quantities, generally interlocked with the quartz. A small amount of rhodonite was also present.

At the beginning of the investigation the only available method for concentrating these ores seemed to be gravity concentration. However, research showed that high intensity magnetic concentration and flotation were applicable.

Gravity concentration. - Gravity concentration of this carbonate ore crushed through 10-mesh was only partly successful. The specific gravity of rhodochrosite is 3.6, and that of quartz is 2.65. The relative densities indicate that a separation should be obtained. However, even with classified feed, tables gave mediocre results with high-silica concentrates. One obstacle of gravity concentration is the fine dissemination of the quartz in the rhodochrosite and the interlocking of the sulphides and quartz. The latter condition is very troublesome because some composite grains have the same density as rhodochrosite. Gravity concentration does not appear promising as a method of treatment for the production of low-silica concentrates. It does, however, offer possibilities by removing some relatively clean quartz at table size, thus eliminating the necessity of grinding all of this resistant mineral to a fine size. Tests show that approximately 55 percent of the quartz can be eliminated by tabling 10-mesh material with a loss of only 12 percent of the manganese.

Magnetic concentration. - A search of the literature on magnetic concentration is not likely to lead one to try the process on rhodochrosite. Nevertheless, the sample examined was found to be decidedly permeable under a high-intensity magnet. Later on, the results of a test in a commercial laboratory on ore crushed through 20-mesh without the customary preliminary sizing showed that 32.1 percent of the ore could be obtained in a concentrate containing about 43 percent manganese and 6.5 percent silica. An additional recovery of manganese could be made by regrinding the middling product. The greater portion of the lead and zinc sulphides in the ore passes into the tailings. Experiments showed that rhodonite had almost the same magnetic permeability as rhodochrosite. Hence, magnetic separation of the two minerals is not possible. Ore containing any appreciable quantity of rhodonite therefore cannot be used,

as the rhodonite passes into the concentrate and makes a siliceous product. If rhodonite is absent a high-intensity magnetic process will undoubtedly give a concentrate which after calcination will be suitable for metallurgical purposes. The objection remains, however, that the ore must be dried and sized before passing over the magnetic separator. The cost of such separators is relatively high, and the amount that each machine could handle would probably not average over a ton of feed per hour. The lead and zinc in the ore would be segregated in the tailing and could be recovered only by re-treating them by some other process.

Flotation. - In the course of the investigation a flotation method that resulted in an excellent separation between the various constituents of the ore was developed. The major part of the sulphides is readily recovered by the use of a small amount of xanthate, copper sulphate, and pine oil. The rhodochrosite is then floated by the addition of sodium oleate or oleic acid and pine oil. The quartz and rhodonite pass into the tailings.

The sulphide concentrates from a typical run had 3.0 percent manganese, 21.3 percent iron, 18.4 percent zinc, and 8.1 percent lead. Less than 1 percent of all the manganese was entrained in the sulphide concentrates.

The finished manganese concentrates made subsequently contained 67.7 percent of all the manganese and assayed 41.55 percent manganese. The tailing had less than 5 percent manganese and contained 10.8 percent of the total manganese in the sample. The middling had 27.35 percent manganese and represented 31.7 percent of the total manganese in the feed. In plant operation the amount of manganese recovered in the concentrates may be increased by part of the manganese in the middling; individual tests have shown that this product is amenable to further cleaning. The finished concentrates contained 6.9 percent insoluble matter, while the tailing included 89.8 percent insoluble. This result shows a decided selectivity. The results of this test and the reagents used are given in table 20.

Since the rhodonite passes into the tailings along with the quartz, manganese ores containing rhodonite can be concentrated readily by flotation.

The practicability of flotation for this ore has been proved by plant tests by the Anaconda Copper Mining Co. A summary of the results at this plant is taken from a current technical publication (84):

Testing of flotation of rhodochrosite ore from the Emma mine of Butte Copper & Zinc at the plant of Anaconda Copper Mining at Anaconda, Mont., indicates that the method can be satisfactorily applied to this manganese-bearing ore. Since December 16, 1929, about 12,000 tons of rhodochrosite ore, averaging about 30 percent manganese and more than 20 percent silica, has been treated. The concentrate has averaged 38 to 40 percent manganese and less than 6 percent silica. The ore is now being treated at the rate of 300 tons daily, and about 200 tons of concentrate is being shipped to the Domestic Manganese & Development plant at Butte. This represents a recovery of about 95 percent of the manganese content of the ore. The nodulized product averages 60 percent manganese and 7 percent silica.

TABLE 20. - Flotation Test of Siliceous Carbonate Ore

Product	Weight, percent	Assay, percent					Percent, total				
		Mn	Fe	Insol.	Zn.	Pb.	Mn	Fe	Insol.	Zn.	Pb
Sulphide concentrate...	5.9	3.05	21.3	16.8	18.4	8.1	0.3	47.6	2.0	74.9	95.5
Manganese concentrate...	33.0	41.55	--	6.9	--	--	67.5	--	4.6	--	--
Middlings.....	15.4	27.35	--	20.8	--	--	20.8	--	9.9	--	--
Tailings.....	45.7	4.78	--	10.8	--	--	10.8	--	83.5	--	--
Composite.....	100.0	20.03	--	49.2	--	--	100.0	--	100.0	--	--
Heads.....	--	21.23	2.54	49.4	1.47	0.50	--	--	--	--	--

Pound per
ton of ore

Reagents used

Sodium ethyl xanthate.....	0.2
Copper sulphate.....	.5
Crude sodium oleate.....	1.0
Pine oil.....	.4

W. B. Daly (61) manager of mines of the Anaconda Copper Mining Co., states that working out a flotation process for treatment of rhodochrosite ores in his company's mines has trebled the reserves of the Butte district. He also states that the cost of the flotation method will not exceed \$1.50 per ton and probably will be \$1.00. The value of the byproducts, lead, zinc, and silver, according to daly, more than offsets half of the milling cost.

Philipsburg District, Mont.

Location and Extent

The Philipsburg district 50 miles northwest of Butte is the chief producer of chemical-grade manganese ores in this country. Over a period of years this district has produced approximately 95 percent of the country's ores of this grade. Although a small tonnage of ore was shipped as early as 1900 substantial production did not begin until about 1916. During 1917 and 1918 the district produced 185,842 tons of ore, mostly for metallurgical purposes, which constituted approximately half the production of the entire country. During this period manufacturers of dry batteries used this ore in the absence of foreign ores, even though the percentage of available oxygen was less. The use of Philipsburg ore has therefore continued.

For the last several years the production of chemical ore from the district has been approximately 15,000 tons per year. Ore of this type commands substantial premiums over metallurgical ore, and more expensive concentration methods may be used to bring it up to grade. The value of the chemical ore shipped during the last few years has been approximately \$40 per ton at the mine.

The Philipsburg district has been described in detail in a report by Emmons and Calkins of the United States Geological Survey (82).

The deposits of manganese ore that have been worked in the district are all in the oxidized zone. The oxidation is complete to a depth of 450 feet and probably extends to a maximum of 750 feet. Carbonate ore is believed to occur in large amounts in the unoxidized zones.

The general nature of the ores in the oxidized and unoxidized zones has been described in detail by Pardee (299, pp. 122-130). He states that the oxide ore consists of pyrolusite, together with smaller amounts of psilomelane, manganite, wad, and braunite. The gangue is mainly quartz, with some kaolin and smaller amounts of calcite and iron oxides. The unoxidized ore consists of a dense aggregate of rhodochrosite, with more or less quartz and a little pyrite or other sulphides.

Some high-grade oxide ores of shipping grade have been removed by selective mining, but in general it may be said that most of the ore requires concentration. The average ore as mined contains 30 to 38 percent manganese and 20 to 30 percent silica. The iron content of these ores is low, generally between 1 and 3 percent. Phosphorus is also low. The ores usually contain a small amount of silver, and although lead and zinc are sometimes found in small quantities the amount of copper is negligible.

Concentration Plants

A gravity mill employing jigs and tables was in operation in this district during 1918 but has since been abandoned. The method now in use for producing chemical-grade ore employs high-intensity magnetic separators. Two mills, the Trout and the Moorlight, have a daily capacity of approximately 100 tons apiece. Details of plant operation at both mills have been described in current publications (28, 96).

The flow sheet at both plants is essentially the same. At the Trout mill, which treats ore from the company mine, the run-of-mine ore is crushed to pass a 1½-inch grizzly. The crushed ore is conveyed to a Ruggles-Cole-type drier 30 feet long and 5 feet in diameter. The fuel used is nut-size coal. The drying operation reduces the moisture content of the ore 12 to 3 percent. The dust from the drier is caught in an 84-inch Cyclone dust collector. The dried ore is carried by a bucket elevator to a Leahy No-blind screen with 1/2-inch openings. The oversize is crushed by a set of 30-by-14-inch rolls operating in closed circuit with the screen. The screen undersize is ground to 6-mesh in a second set of 30-by-14-inch rolls, operated in closed circuit with a Colorado impact screen. This minus 6-mesh ore is sized further by a second Leahy screen into a plus 14-mesh product and a minus 14-mesh product. The sized ore then passes to storage bins of 200 tons capacity above the magnetic separators, where it is fed to three type 3-E Wetherill separators. Each separator has three magnets wound respectively with 30,000, 60,000 and 100,000 ampere turns and is operated with a direct electric current of about 120 volts. Concentrates are discharged into bins, whence they can be conveyed directly into railway cars.

Dust from dry crushers and screens is collected by housings connected with a second 84-inch Cyclone dust collector, placed over elevators and screens. This dust, and that collected during the drying operation, is wetted and tabled on a Plat-O slimer for a high-grade concentrate. Tailings from this operation are impounded, and those from the magnetic separators are placed on a storage dump. According to Furness (104, p. 188).

The ore treated contains from 30 to 35 percent manganese and 25 to 30 percent silica. Two and one-quarter tons of crude ore were required to make one ton of product containing 70 percent of manganese dioxide, less than 0.2 percent of phosphorus, less than 6 percent of iron, and approximately 10 percent of silica.

The flow sheet at the Moorlight mill, a custom mill, is much the same as at the Trout mill, except that the ore before being passed to magnetic separators is screened to produce 5 sizes instead of 2. At this plant no attempt is made to save the dust. The tailings are impounded.

The magnetic process as outlined yields a desirable product for the dry-battery industry. Tailings from the magnetic process contain about 20 percent manganese. Some of the ores contain impurities which would make them unsuitable for chemical purposes, and while they would be suitable for metallurgical purposes the lower value of this class of ore would not permit the treatment cost of a magnetic process.

Geologists who have examined the district agree that deposits of rhodochrosite ore, probably of large size, lie below the zone of oxidation. Treatment of such material constitutes a problem for the future.

Concentration Tests

In general it may be said that the higher-grade ores of the district can be adequately concentrated by the magnetic process now in use. The concentration problems of Philipsburg may therefore be said to deal with the following three classes of material:

- (1) Oxide ores too low-grade to justify the use of the magnetic process and ores containing impurities which make them unfit for chemical purposes.
- (2) Tailings from previous or present metallurgical processes.
- (3) Unoxidized or carbonate ores.

During the investigation many samples of manganese oxide from Philipsburg material were examined. These samples included ores now being treated by the magnetic process, mill products, including tailings, now going to waste, low-grade dump ore, and ore not suitable for chemical purposes. Tests were made to determine the amenability of the various ores to gravity, magnetic, and flotation methods of separation.

Ore Unsuitable for Chemical Grade. - A sample containing approximately 35 percent manganese, 24 percent silica, and 4.5 percent lime, which on account of its physical and chemical characteristics was not suitable for chemical purposes, was examined. The manganese minerals in the ore consisted mainly of pyrolusite and psilomelane, and the gangue was mostly quartz. Microscopic examination showed that the manganese minerals were locked with the silica and that the production of a low-silica concentrate at a coarse size by any ore-dressing process was unlikely. Float-and-sink tests of this ore, followed by table tests, indicated that if it were crushed to 10-mesh, classified, and tabled, concentrates containing approximately 44 percent manganese and 12 percent silica could be made. Flotation tests gave unsatisfactory results. Pyrolusite is the most readily floatable of the oxide-manganese minerals, and as float-and-sink tests showed that manganese minerals upon grinding to 100-mesh had locked with them less than 7 percent silica, favorable results were therefore expected. A detailed study of the problem brought out the fact that the ore contained a large percentage of soluble salts which seriously interfered with the selectivity of the process. For this reason an all-flotation process on the oxide ores does not seem possible, although it is very probable that flotation would adequately handle the slime produced from table operations because much of the soluble salts would be washed out of the ore during crushing, tabling, and dewatering.

Tests of some of the low-grade ores indicate that their silica content can be reduced by gravity concentration. It is doubtful if the silica content can be reduced to the 8-percent limit required to make ferromanganese. Concentrates containing 12 to 15 percent silica can be made, and while such

concentrates are not in demand now they could be used in emergency. The ratio of manganese to iron is satisfactory, and the phosphorus content seldom exceeds 0.15 percent.

Tailings. - During mining and milling dumps of sorted low-grade ore and mill tailings have accumulated which probably total 100,000 or 150,000 tons. The average analysis is not accurately known, although the tailings from the magnetic plants are said to contain 20 to 22 percent manganese. Samples from one dump contained about 39 percent manganese and 25 percent silica.

In 1927 some of the higher-grade dump ore was shipped to Utah, where it was used in the production of manganiferous pig iron. During 1927, 1928, 1929, and 1930 shipments of ferruginous manganese ores from Montana have been 10,332, 11,156, 15,665, and 11,875 tons, respectively, by years. Some of the higher-grade dump ore undoubtedly can be used in this way, but much of it is too low-grade to be marketable without concentration.

Examination of one of the higher-grade dump ores which contained 38.9 percent manganese, 1.6 percent iron, 26.6 percent insoluble, and 0.054 percent phosphorus showed that gravity concentration could grade this ore up to about 46 percent manganese and 14 percent silica.

Because these tailings are continually accumulating their treatment presents a problem. The samples studied had about 18 percent manganese, 2.1 percent iron, 58.4 percent insoluble, and 0.176 percent phosphorus. The liberated high-grade manganese minerals had been removed by the magnetic process, and the manganese minerals remaining in the tailings were badly locked with gangue. Float-and-sink tests, as well as high-intensity magnetic concentration of 20- to 35-mesh tailings, indicated that concentrates from either method would contain over 25 percent of silica. Finer grinding would undoubtedly result in somewhat cleaner concentrates.

According to Fitzberg (96), tailings from the Trout mill can be ground and re-treated by the magnetic process with an increase in the over-all recovery from 80 to 95 percent. However, manganese ore suitable for battery purposes must contain not more than a certain percentage of fines, and difficulty would be experienced in finding a market for such material. It seems probable, therefore, that the tailings will have to be ground so fine that they will be unsuitable for battery use. By sintering, however, these magnetic concentrates could be used for metallurgical purposes. Tabling of these tailings is not promising due to the fine grinding required and the resulting large slime loss. Flotation offers a possibility of treatment, although the presence of soluble salts has made results to date rather unsatisfactory.

Unoxidized Ores. - The deposits of manganese-bearing ores lying below the oxidized zone have not been opened to any extent, and samples of such material were not obtainable for tests.

Las Vegas District, Nev.Location and Extent

In 1917 deposits of metallurgical manganese ores were discovered 16 miles east of Las Vegas, Nev., and about 5 miles west of the Colorado River. The discovery mine, the Three Kids, produced 15,045 tons of manganese ore in 1917 and 1918, but since that time the total production has amounted to less than 1,000 ton (104, p. 138). Due to consolidation of the Three Kids and Las Vegas groups the deposits of this area again attracted attention in 1929.

Mineralogy

The manganese-bearing deposits occur in low hills composed of volcanic flows and tuffs, gypsum, sand, and clay, which border the Colorado River. The manganese deposits have been formed by partial replacement of the tuffs and the sand and clay beds. The ore is a dark-brown fine-grained porous material of low specific gravity. Manganese is present mainly as wad, although small amounts of pyrolusite and occasionally psilomelane are found.

Concentration Tests

Records show that the ore shipped from the Three Kids mine contained 39 percent manganese, 11 percent silica, 1 percent iron, 0.3 percent phosphorus, and 10 percent moisture. The average grade of ore remaining in the district is leaner and requires concentration.

From the Las Vegas group of claims 12 samples requiring concentration were examined. These varied considerably in grade but had the same general character. Concentration tests of a composite sample only are reported.

This material had a tenor of approximately 24 percent manganese and 35 percent silica. The iron content was about 2 percent, and the phosphorus was low - 0.05 percent. This ore-bearing material consists of a pink clay and tuff which has been partly replaced by soft black amorphous manganese oxides, largely wad. The ore is quite porous and of light weight. At first appearance some of the ore seems fairly high-grade, but more thorough examination under the microscope shows that the original beds of clay and tuff have been only partly replaced by the manganese oxides. The amount of clean manganese mineral grains of any appreciable size was small. This preliminary examination was checked by float-and-sink tests which corroborated the evidence that the material would be difficult to concentrate.

The poor liberation of the gangue prevented concentration at 1/2 inch. Gravity concentrates of material sized between 1/2 inch and 10-mesh had a silica content of about 30 percent. When the ore was stage-crushed to 10-mesh approximately one third was finer than 100-mesh. Float-and-sink tests in table 21 show that gravity concentrates contain about 25 percent silica. The locking of the manganese and gangue in sizes as fine as 100-mesh and the presence of soluble salts prevented good flotation.

TABLE 21. - Float-and-Sink Tests of Las Vegas Ore

Mesh	Product	Weight, percent	Assay, percent			Percent of total		
			Mn	Fe	SiO ₂	Mn	Fe	SiO ₂
8 to 28.....	Float on 2.95..	25.2	12.72	2.3	51.20	11.8	26.1	40.3
	Sink in 2.95...	74.8	32.17	2.2	25.56	88.2	73.9	59.7
	Total.....	100.0	27.27	2.2	32.02	100.0	100.0	100.0
28 to 100...	Float on 2.95..	45.5	6.78	2.2	62.75	14.8	41.7	67.4
	Sink in 2.95...	54.5	32.66	2.6	25.30	85.2	58.3	32.6
	Total.....	100.0	20.88	2.4	42.34	100.0	100.0	100.0

Franklin Furnace, N. J.

The ores of Franklin Furnace, N. J., are mined primarily for their zinc content. They do, however, contain an appreciable amount of manganese, which is largely recovered in the residues from zinc smelting. These residues are smelted, and from them a large share of the country's spiegeleisen is produced.

The chief zinc minerals in these ores are franklinite, willemite, and zincite. Of these franklinite (Fe, Zn, Mn) O. (Fe, Mn)₂O₃, which contains 10 to 17 percent manganese, is most abundant. Concentration at the New Jersey Zinc Co. plant consists of removing the permeable franklinite by a magnetic process and recovering the other zinc minerals by gravity concentration. Smelting of the franklinite concentrates yields a residue having a composition similar to a ferruginous manganese ore.

The major part of the residues contains 12 to 14 percent manganese and approximately 40 percent iron. Material of this grade is used for the manufacture of spiegeleisen. The smaller part of the residues contains about 7 percent manganese and is used in the manufacture of pig iron. Statistics (104) for 1926 show that 74,096 tons of spiegeleisen were made from 149,374 tons of zinc residues containing 13 percent manganese. In this same year 11,219 tons of low-grade residues containing approximately 7.5 percent manganese were used in the manufacture of pig iron.

During the last 5 years an average of about 140,000 tons of zinc residues has been shipped from Franklin Furnace annually. Spiegeleisen has been made from this since 1870, and records of the material shipped from 1889 to 1930 show a production of approximately 4,000,000 tons.

Inasmuch as the manganese from this district is a byproduct in the smelting of zinc ores, speiegeleisen can be produced at reasonable cost. These ore deposits apparently have a long life, and it is believed that an annual production of approximately 75,000 tons or more of speigel can be expected for many years. An additional smaller production of ore suitable for the manufacture of manganiferous pig iron may also be expected.

Little Florida Mountains, N. Mex.Location and Extent

During recent years New Mexico has become of increasing importance as a producer of manganese ore. The greater part of the production has come from the Manganese Valley mine, on the east flank of the Little Florida Mountains, Luna County, about 16 miles east of Deming.

Associated with the high-grade metallurgical ores are lower-grade ores, some of which are being successfully concentrated. In addition to the ore body now being worked, somewhat similar deposits are found elsewhere along the same fissure.

The manganese-bearing ores occur in a fault fissure in agglomerate on the east side of the Little Florida Mountains, which are made up of agglomerate and felsitic rhyolite. The geology of the district is described by Furness (104, p. 120).

Mineralogy

Several types of ores of varying mineralogical content and structure are found. The manganese minerals are psilomelane, pyrolusite, manganite, and wad. The most important gangue constituent is a rhyolite, and the type of ore varies with the degree to which it has weathered. In some ores the rhyolite has decomposed to such an extent that little crushing is needed to give liberation. Other ores consist of a hard, unaltered breccia requiring fine grinding to free the manganese minerals.

Manganese Valley Mill

Ore mined from the Manganese Valley mine is trammed to the mill and dumped over a grizzly where the lump ore and waste are picked. Approximately 1 ton of lump ore is obtained from every 6 tons mined. The grizzly undersize constitutes the mill feed and contains about 22 percent manganese. Seven sizes are made, and by the use of storage hoppers the flow sheet is so adjusted that three jigs suffice. A representative analysis of 4,000 tons of concentrates shipped from this mine, according to Furness (104, p. 192), is as follows: Manganese, 45.75 percent; silica, 2.50 percent; iron, 2.0 percent; aluminum, 0.50 percent; copper, 0.078 percent; lead, 0.051 percent; calcium oxide, 5.50 percent; nickel, none; phosphorus pentoxide, trace; magnesium oxide, trace.

Concentration Tests. - No great difficulty has been experienced at the mill of the Manganese Valley mine in producing concentrates of satisfactory grade. The loss of manganese in the fines, however, has been considerable, although during the past year this trouble has been partly solved.

Some milling ore in the district is not so free from locked particles as that treated in the aforementioned mill, and concentration of such material will be more difficult. Particular attention was therefore given to the fine material lost by the method of concentration in use and to concentration of

the so-called "frozen ores" -- those requiring considerable crushing to free the manganese minerals from the gangue.

Until the early part of 1929 ores finer than one-fourth inch were not treated but were simply stock-piled, and at that time approximately 15,000 tons of such fines had accumulated. A representative sample was examined to determine whether it could be concentrated. The examination showed that much of the manganese was present as pyrolusite and psilomelane, with some wad. The gangue constituents were rhyolite, with some calcite. There was little locking of the manganese and gangue minerals.

Float-and-sink tests were made of various sizes to determine the grade of concentrates that could be produced by jigging or tabling. The results, given in table 22, show that with good operating conditions over 80 percent of the manganese should be recovered in the form of concentrates containing over 40 percent manganese and less than 6.0 percent silica.

TABLE 22. - Float-and-Sink Tests of Stock-Piled Mill Fines

Size, mesh	Product	Weight, percent	Assay, percent			Percent of total		
			Mn	Fe	Insol.	Mn	Fe	Insol.
1/4 inch to 10.	Float on 2.92..	53.1	4.6	2.2	68.6	11.3	48.9	92.8
	Sink in 2.92...	46.9	40.9	2.6	6.0	88.7	51.1	7.2
	Total.....	100.0	21.6	2.4	39.3	100.0	100.0	100.0
10 to 35.....	Float on 2.92..	54.8	6.4	2.1	59.8	15.1	48.5	92.9
	Sink in 2.92...	45.2	43.7	2.7	5.5	84.9	51.5	7.1
	Total.....	100.0	23.3	2.4	35.3	100.0	100.0	100.0
35 to 65.....	Float on 2.92..	53.8	4.8	2.3	64.7	12.6	37.3	89.8
	Sink in 2.92...	46.2	38.7	4.5	8.6	87.4	62.7	11.2
	Total.....	100.0	20.5	3.3	38.8	100.0	100.0	100.0

The minus 1/4-inch material is now being treated at the Manganese Valley mine. It is deslimed in a spiral conveyor, and the coarse material is jigged. From the sizing analysis of the fines recovery probably could be increased by jigging the size coarser than 8-mesh and classifying and tabling the minus 8-mesh material.

Some fine pyrolusite will be lost, even by such a system, due to the tendency of this material to slime. Tests were accordingly made of a sample which broke readily and gave low recoveries in the mill due to slime losses. The chief manganese mineral in this sample was pyrolusite, with smaller amounts of psilomelane. The gangue constituent is a felsitic rhyolite.

Preliminary to gravity-concentration tests, two sizes, (1/2-inch to 10-mesh and 10- to 65-mesh) were fractionated with a heavy liquid having a specific gravity of 2.93. The results given in table 23 show that the ore is well-

liberated at the sizes tested. Actual tabling tests of ore crushed to 10-mesh showed a recovery of only 60 percent of the manganese due to the large loss of manganese in the slime.

A flotation test, using the reagents listed on page 60, recovered a concentrate with a tenor of 45.6 percent manganese, 2.8 percent iron, and 9.2 percent insoluble. The percentage of manganese recovered was 64.5 percent. The middlings were not re-treated.

As has been mentioned, some of the ore in the district is more difficult to treat than that milled at the Manganese Valley mill. A sample of such ore from an ore body north of the present workings was examined. It was brecciated, and the gangue constituents were a felsitic rhyolite with a small amount of barite. The manganese occurs mainly as psilomelane, although some pyrolusite is also present. The specific gravity of the psilomelane is approximately 4.2 and that of the rhyolite 2.6.

TABLE 23. - Float-and-Sink Tests of Soft Ore

Mesh	Product	Weight, percent	Assay, percent			Percent of total		
			Mn	Fe	Insol.	Mn	Fe	Insol.
1/2 inch to 10.	Float on 2.93..	42.5	2.1	3.7	88.6	3.2	31.8	95.9
	Sink in 2.93...	57.5	46.5	5.9	2.8	96.8	68.2	4.1
	Total.....	100.0	27.6	4.9	39.5	100.0	100.0	100.0
10 to 65.....	Float on 2.93..	34.5	1.7	2.7	86.3	2.1	15.4	87.3
	Sink in 2.93...	65.7	42.3	7.8	6.5	97.9	84.6	12.7
	Total.....	100.0	28.4	6.0	33.9	100.0	100.0	100.0

Visual examination of the ore indicated that crushing to one-half inch would be required to liberate the gangue minerals.

Gravity concentration of this ore will be somewhat more difficult than that of the ores now being treated, as indicated by float-and-sink tests of ore sized between 1/2-inch and 10-mesh and from 10- to 35-mesh. Material coarser than 10-mesh yielded a sink product containing 13.4 percent of insoluble when fractionated with a heavy liquid of 2.95 specific gravity. These results are given in table 24.

The dissection of the 10- to 35-mesh size showed the presence of 10 percent of manganese in locked mineral of a specific gravity from 2.95 to 3.5. The heavy liquid fraction heavier than 3.5 contained 11.5 percent insoluble. The recovery of manganese is high, while the grade of concentrates is only fair.

The brecciated ores will be more difficult to treat than those now being mined, and the manganese content of the concentrates will be lower because of the presence of more locked gangue and also because of some associated barite. The percentage of iron and phosphorus in the concentrates will, however, be low.

TABLE 24. - Float-and-Sink Tests of Hard Ores

Mesh	Product	Weight, percent	Assay, percent			Percent of total		
			Mn	Fe	Insol.	Mn	Fe	Insol.
1/2 inch to 10.	Float on 2.95..	45.6	4.1	2.4	84.9	7.7	60.5	84.1
	Sink in 2.95...	54.4	41.15	1.3	13.4	92.3	39.5	15.9
	Total.....	100.0	24.3	1.8	46.0	100.0	100.0	100.0
10 to 35.....	Float on 2.95..	47.0	2.4	1.9	88.2	5.7	46.4	80.2
	2.95 to 3.2....	6.9	8.5	1.7	58.9	3.0	5.7	7.9
	3.2 to 3.5.....	3.3	13.7	1.7	37.1	2.3	3.1	2.4
	Sink in 3.5....	42.8	41.3	2.0	11.5	89.0	44.8	9.5
	Total.....	100.0	19.9	1.9	51.7	100.0	100.0	100.0

Silver City District, N. Mex.Location and Extent

Large shipments of ferruginous manganese ore during and since the war make the Silver City district of interest. Only a small quantity of high-grade manganese ore has been shipped from this area, but much ore suitable for the production of spiegeleisen has been mined.

Production figures of New Mexico, taken largely from Mineral Resources of the United States, show that from 1913 to 1929 the State produced roughly 350,000 tons of ferruginous manganese ore, the greater portion of which came from the Silver City district.

The most important deposits are on Boston Hill near Silver City. The deposits have been described by Wells (386) and by Goodier (120).

Mineralogy

The ore consists of an intimate association of iron and manganese oxides, occurring as fissure fillings and replacements in limestone. The chief ore minerals are pyrolusite and manganite, with some limonite and hematite. The ore ranges in color from dark red to black and is also of varying structure and hardness. Some of it is extremely hard and brittle, although some crumbles readily. Nodular ore is found in certain parts of the ore body. The iron and manganese minerals are closely intergrown and require fine grinding for liberation.

Concentration Tests

Samples of a ferruginous manganese ore and a siliceous ore were examined. Experiments were made with the ferruginous manganese ore to determine whether a high-manganese concentrate suitable for the manufacture of ferromanganese could be produced. The siliceous ore was examined to determine whether enough silica could be eliminated to make it marketable.

Ferruginous Manganese Ore. - An ore having the following analysis was examined: Iron, 33.67 percent; manganese, 16.81 percent; insoluble, 7.80 percent; silica, 5.48 percent; and phosphorus, 0.01 percent.

The ore was given a reducing roast at 550°C. for 30 minutes, then divided into three portions crushed to pass screens having 8, 28, and 100 meshes, respectively, per linear inch. Each size was then separated into a magnetic and a nonmagnetic portion by means of a Davis magnetic tube. The results, given in table 25, show the intimate association of the manganese and iron oxides. Even after grinding to 100-mesh, 43 percent of the manganese still remained locked with enough iron to render the mixture magnetic. The ratio of iron to manganese in all the nonmagnetic portions is too high to permit the material to be used in the manufacture of ferromanganese. Finer grinding of the roasted ore before concentration would undoubtedly increase the percentage of manganese liberated. Such fine grinding, however, in addition to the added cost would be undesirable in that it would produce a more siliceous manganese concentrate, since all the silica not locked with the iron particles would pass into the nonmagnetic product with the manganese minerals. The roasting and magnetic concentration process for treating such ores, therefore, is not promising.

The ore shipped to date has been of direct shipping grade, none having been concentrated. The average tenor of the ore from one of the larger mines, as given by Goodier (120), is reported as being 39 percent iron, 13 percent manganese, and 8 percent insoluble, the balance being largely lime. The phosphorus content is low, averaging about 0.012 percent. Estimates made in 1917 by Umpleby (374), when development work had not reached more than 40 feet below the surface, gave the district an ore reserve of at least 500,000 tons and possibly 1,000,000 tons of ore similar to that already mined.

Siliceous Ore. - The siliceous ore was similar to the ferruginous manganese ore, except that more free gangue was present. Although the ore bodies occur in a limestone formation the gangue mineral directly associated with the economic minerals in the low-grade ore looks quite unlike limestone. Apparently most of the original limestone has been replaced by silica and iron oxide, so that it has an appearance and chemical composition similar to a ferruginous chert. Visual examination of the ore indicated that the greater portion of the gangue would be liberated by crushing to $\frac{1}{2}$ -inch.

Float-and-sink tests of the ore sized between $\frac{1}{2}$ -inch and 10-mesh and 10- to 65-mesh were made. These separations, made with a heavy liquid with a specific gravity of 3.30, indicated that by jigging the coarser sizes and tabling the finer sizes concentrates containing about 36 percent iron, 12.50 percent manganese, and 9.2 percent silica could be made. The results are given in table 26.

Chamberlain District, S. Dak.

Location and Extent

Deposits of manganiferous material were discovered along the Missouri River near Chamberlain, S. Dak., in 1928. Because of their large size these deposits, although low in grade, have attracted much attention. Reports on the occurrence of manganese ore in this district are to be found in the current literature (83, 376).

TABLE 25. - Roasting and Magnetic Tube Test of Ferruginous Manganese Ore

Size, mesh	Product	Weight, percent	Assay, percent					Percent of total				
			Mn	Fe	Insol.	SiO ₂	P	Mn	Fe	Insol.	SiO ₂	P
Minus 8	(Concentrate	85.0	13.57	46.23	7.62	- -	- -	66.9	97.6	83.3	- -	- -
	(Tailing ...	14.0	41.32	7.04	9.36	- -	- -	33.1	2.4	15.7	- -	- -
	Composite	100.0	17.45	40.75	7.86	- -	- -	100.0	100.0	100.0	- -	- -
Minus 28	(Concentrate	81.7	12.82	47.52	7.46	- -	- -	60.4	96.9	69.6	- -	- -
	(Tailing ...	18.3	37.41	6.84	14.56	- -	- -	39.6	3.1	30.4	- -	- -
	Composite	100.0	17.32	40.07	8.75	- -	- -	100.0	100.0	100.0	- -	- -
Minus 100 ...	(Concentrate	72.4	11.13	52.87	5.72	4.30	0.010	43.2	94.3	48.1	50.65	50.0
	(Tailing ...	27.6	36.40	8.33	16.16	10.91	.027	56.8	3.7	51.9	49.35	50.0
	Composite	100.0	18.65	40.58	8.60	6.14	0.014	100.0	100.0	100.0	100.0	100.0

TABLE 26. - Float-and-Sink Tests of Siliceous Ore

Mesh	Product	Weight, percent	Assay, percent			Percent of total		
			Mn	Fe	SiO ₂	Mn	Fe	SiO ₂
1/2 inch to 10.	Float on 3.30..	40.7	3.51	13.73	52.50	16.3	20.8	79.9
	Sink in 3.30...	59.3	12.40	35.88	9.08	83.7	79.2	20.1
	Total.....	100.0	8.78	26.87	26.75	100.0	100.0	100.0
10 to 65.....	Float on 3.30..	38.5	2.84	10.01	59.85	12.2	14.4	79.5
	Sink in 3.30...	61.5	12.71	37.31	9.68	87.8	85.6	20.5
	Total.....	100.0	8.91	26.80	28.99	100.0	100.0	100.0

Manganese in this district is found as manganiferous iron carbonate nodules scattered through a bed of shale 35 to 40 feet thick. The manganese content of the bed of shale as a whole will average less than 2 percent. The nodules will average about 16 percent manganese and 8 percent iron.

According to the Interior Department press release just cited, the average yield of nodules from each yard of excavated material from 12 shafts on the property was 164 pounds. These nodules had an average analysis of 15.70 percent manganese and 11.08 percent iron; the phosphorus content is high, ranging from 0.38 to 0.55 percent; the silica and alumina contents were approximately 13 and 2.5 percent, respectively. A more detailed analysis of these nodules shows them to contain approximately 16 percent lime, 1.6 percent magnesia, and 0.05 percent sulphur, with a loss of about 25 percent on ignition.

Mineralogy

The manganese-bearing nodules consist of an isomorphous mixture of rhodochrosite, calcite, magnesite, and siderite. Fine grinding has given no indication of liberating the respective minerals. The specific gravity of the nodules is approximately 3.0 and the hardness about 3.5.

Concentration Tests

The dissemination of the minerals in the nodules as noted above was such that a brief examination showed ore dressing would be futile. Samples of the original material, consisting of the nodules in the shale, were not obtained, and the ease of separation of these nodules could not be tested.

The nodules are separated from the shale very easily by mere drying of the broken material in air for a few days. The procedure followed was that of breaking the ore bank, permitting it to stand a few days, and screening out the nodules which were then free from the shale. The manganese recovery by this procedure was very high (97.9 percent total). The surface of the deposit was covered with free nodules.

Northeast Tennessee DistrictLocation and Extent

Numerous manganese mines and prospects are found in eastern Tennessee. They lie in a zone which extends diagonally across the state near its eastern border. Stose and Schrader (359), who examined the deposits in 1918, state that eastern Tennessee as a whole contains 130 or more mines or prospects. Although many deposits are known they are generally small, and no one has produced a large tonnage. Records from Mineral Resources of the United States show that the production of manganese ore from 1886 to the end of 1929 has been approximately 14,000 tons, and the production of ferruginous manganese ore during the same period about 5,000 tons. The district as a whole produced 6,206 tons of manganese ore and 2,158 tons of ferruginous manganese ore in 1917 and 1918. Since the war the annual production of high-grade ore has averaged less than 500 tons.

The chief manganese-producing area in eastern Tennessee, and the one that seems to have the greatest future, is in Johnson and Carter Counties, in the northeastern corner of the State. This discussion will therefore be devoted to the ores from this area. The deposits in these two counties are in the form of oxides scattered through residual clay or residual sand derived from the weathering of sedimentary rocks. According to Stose and Schrader (359):

***Most of the oxide ores may be classed as replacement deposits, some of them replacing the rock in place, but most of them replacing clay and sand, residual from rock decay, which still retain the rock structure. None of them except small amounts of ore associated with other deposits are believed to be strictly residual detrital deposits concentrated by gravity.

In this area the ore is found in residual deposits formed through the decay of the Shady dolomite and the Wantaga shale, (the first formation predominating.)

Mineralogy

Manganese is found in these deposits chiefly as psilomelane, which occurs as dense nodules, irregular fragments, and porous particles. The mineral is disseminated through clay. In some places, notably at the Cedar Hill mine, pockets of high-grade pyrolusite are found. Manganite and wad are also present in some of the deposits. Limonite is common in many of the ore bodies. Some of the limonite is in clean grains, but much of it is disseminated through the manganese particles. Gangue constituents associated with the clay are aggregates of quartz, weathered chert, and particles of sandstone.

Concentration Tests

Almost all the ore of the district requires some form of concentration. In the past, log washers were used to remove the clay; the log-washer concentrates were shipped without further concentration. For a few ores such a process makes a suitable product, but in the majority of the deposits the presence of chert and quartz, which are not removable, makes the log-washer concentrates too high in silica.

Gravity Concentration. - Examination indicated that the problem of concentration, particularly on ores from Carter County, was largely one of treating log-washer concentrates finer than one-half inch. Much of the clay can be eliminated from the ore by log washing, and fortunately the coarser portion of the ore consists of manganese oxides. Most of the chert and gangue particles are finer than one-half inch. The logical steps in the process are therefore log washing to remove clay, screening of the log-washer concentrates at approximately one-half inch for a coarse concentrate, and further concentration of the minus one-half inch material.

Samples of minus one-half inch washed material from 6 mines were examined in detail, but for the sake of brevity the results of the work on a composite made of samples from 5 of the 6 properties will be reported. An additional reason for so reporting the work is that the size of the individual deposits would not justify a separate mill, and as the mines are not far from each other the practical way to treat these ores would seem to be to log-wash the ore at each mine and send the log concentrates to a central plant for further treatment.

The composite sample had a tenor of 26.3 percent manganese, 12.4 percent iron, 29.0 percent insoluble residues, and 0.25 percent phosphorus.

Float-and-sink tests of sized portions of the ore showed that the mineral particles with a specific gravity greater than 3.2 contain 11.2 to 9.1 percent insoluble residues. The results are shown in table 27.

On first examination the mineral particles in these ores appear clean, but from the results they evidently contain an appreciable amount of very finely divided gangue. To determine the amount of silica in the apparently clean mineral a representative sample of 3-mesh ore was hand-picked; all particles of free gangue and all particles of manganese minerals to which gangue could be seen adhering were discarded. The specific gravities of the remaining apparently clean manganese minerals were determined with a Joly balance, then the particles were combined according to their specific gravities. The results of the assays given in table 28 show that the silica content varies in a general way with the specific gravity; the presence of both iron and manganese and the porosity of the mineral cause some irregularities. The greater portion of the limonitic grains is only slightly lower in specific gravity than the cleanest manganese particles, thus it is obvious that they cannot be removed by jigging or tabling without causing an excessive loss of manganese.

These results have particular interest, because many manganese ores found in residual deposits throughout Virginia and Tennessee show similar characteristics; that is, particles of apparently clean psilomelane vary in specific gravity from little more than that of the gangue to over 4.0 and contain amounts of silica up to 25 percent. With such ores, concentration is difficult, and the operator must choose between a small recovery of a high-grade concentrate or a greater recovery of a siliceous concentrate. Fine grinding of such material has little value, since the gangue particles are so small and so scattered through the manganese oxide that they are not liberated by grinding to 100-mesh.

TABLE 27. - Float-and-Sink Test of Composite Sample,
Minus 1/2-Inch Log-Washer Concentrate

Size, mesh	Product	Weight, percent	Assay, percent			Percent of total		
			Mn	Fe	Insol.	Mn	Fe	Insol.
1/2 inch to 4.	Float on 3.20..	39.2	11.0	17.7	47.4	16.0	48.5	73.2
	Sink in 3.20..	60.8	37.0	12.1	11.2	84.0	51.5	26.8
	Total.....	100.0	26.8	14.3	25.4	100.0	100.0	100.0
4 to 10.....	Float on 3.20..	35.2	9.4	13.0	58.3	11.8	38.5	77.2
	Sink in 3.20..	64.8	37.9	11.2	9.3	88.2	61.5	22.8
	Total.....	100.0	27.9	11.8	26.5	100.0	100.0	100.0
10 to 65.....	Float on 3.20..	45.1	7.8	8.4	70.0	14.5	36.8	86.3
	Sink in 3.20..	54.9	38.0	11.5	9.1	85.5	63.2	13.7
	Total.....	100.0	24.4	10.1	36.5	100.0	100.0	100.0

TABLE 28. - Specific-Gravity Analysis of Hand-Sorted
Concentrate, Through 1/2-Inch

Specific gravity	Weight, percent	Assay, percent			Percent of total		
		Mn	Fe	Insol.	Mn	Fe	Insol.
3.0 to 3.2.....	4.9	0.5	41.7	22.8	0.1	14.0	6.9
3.2 to 3.3.....	9.0	15.6	25.5	24.5	4.3	16.1	13.6
3.3 to 3.4.....	15.0	36.0	12.1	12.8	16.8	12.6	11.9
3.4 to 3.5.....	6.7	.9	38.2	35.6	.3	18.2	15.0
3.5 to 3.6.....	11.8	14.9	28.3	24.9	5.6	23.1	18.2
3.6 to 3.7.....	7.5	42.4	6.1	13.3	9.9	3.5	6.3
3.7 to 3.8.....	30.6	43.1	4.1	12.0	40.9	9.0	23.1
3.8 to 3.9.....	14.5	48.8	3.3	5.5	22.1	3.5	5.0
Composite.....	100.0	32.2	14.3	16.0	100.0	100.0	100.0

The foregoing statements indicate that the minus one-half inch portion of this ore is difficult to concentrate. The coarser portion is, however, clean enough so that the low-grade concentrates obtainable from the finer sizes may be mixed with it and a marketable product produced. Such concentrates would contain too much iron, silica, and phosphorus to be used for ferromanganese but could be used for other metallurgical purposes.

Magnetic Concentration. - Preliminary magnetic tests gave encouragement. The cooperation of the Dings Magnetic Separator Co. of Milwaukee, Wis., was obtained.

A sample of the composite log-washer concentrate was crushed to pass 12-mesh and given several passes through a high-intensity magnetic separator. Either the strength of the magnetic field or the belt speed was varied for each

pass. From the results in table 29 it may be seen that a small proportion of the iron mineral is highly magnetic and can be removed with a low-intensity magnetic field that will not affect the manganese. However, after this small amount of iron, which is mainly magnetite, is removed no clean-cut separation can be effected between limonite and the manganese minerals. Contrary to expectations, the limonite was slightly less magnetic than psilomelane. A good separation was made between the manganese minerals and the free siliceous particles. The silica in the manganese concentrates was locked. The analyses of the magnetic manganese concentrates compared closely to the heavier material obtained by fractionating a like size sample with a heavy liquid of 2.90 specific gravity.

These tests are interesting primarily to show the characteristics of the ore. High-intensity separation has been found practical for the concentration of chemical grade ore in the Philipsburg (Mont.) district, but it has as yet not proved economical for the production of metallurgical ore at present prices.

TABLE 29. - High Intensity Magnetic Test of Composite Sample,
Minus 12-mesh Log-Washer Concentrate

Magnetic intensity	Belt speed, ft. p.m.	Current, amperes	Weight, percent	Assay, percent			Percent of total		
				Mn	Fe	Insol.	Mn	Fe	Insol.
Low.....	75	0.5	1.7	2.4	47.2	20.7	0.2	6.0	1.3
Medium.....	75	1.5	5.5	39.3	9.9	8.8	8.1	4.0	1.8
Medium high	75	2.0	35.0	37.1	11.8	10.7	48.9	30.8	13.7
High.....	75	2.5	23.5	32.4	15.1	13.4	28.7	26.5	11.5
Extreme....	50	2.5	15.0	21.6	20.6	23.2	12.2	23.1	12.8
Nonmagnetic	50	2.5	19.3	2.6	6.7	83.2	1.9	9.6	58.9
Total..	--	--	100.0	26.5	13.4	27.3	100.0	100.0	100.0

Roasting and magnetic tests were made of jig concentrates to produce a high-grade ore suitable for ferromanganese. This gravity concentrate, sized between 1/2-inch and 10-mesh, was given a reducing roast and the ore separated into magnetic and nonmagnetic portions. The results, given in table 30, indicate that a high-manganese, low-iron concentrate can be produced. The phosphorus and silica are shown to be associated more closely with the iron than with the manganese particles. However, the phosphorus content of the manganese concentrate is still high (0.325 percent), rendering the material unusable for the manufacture of ferromanganese.

TABLE 30. - Roasting and Magnetic Test of Minus 1/2-Inch
C Plus 10-Mesh Composite Gravity Concentrate

Product	Weight, percent	Assay, percent				Percent of total			
		Mn	Fe	Insol.	P	Mn	Fe	Insol.	P
Magnetic concentrate...	43.5	23.2	31.5	14.3	0.554	37.5	84.0	50.5	56.7
Nonmagnetic concentrate	56.5	49.1	4.6	10.8	.325	62.5	16.0	49.5	43.3
Composite.....	100.0	37.8	16.3	12.3	.425	100.0	100.0	100.0	100.0

Flotation. - Flotation tests met with some success, although difficulty was experienced in overcoming the ill effects of the slimed iron oxides, which tended to cause dirty concentrates. The results of one test in which the ore was ground to 100-mesh, after which the finest slimes were removed before flotation, are shown in table 31.

Grinding log-washer concentrates for flotation does not seem justified. It is believed that concentrates of satisfactory grade can be made by jigging and tabling at lower cost.

A particular field for the use of the flotation process does, however, exist. Large amounts of fine manganese minerals, largely pyrolusite, are lost in log washing. Where the size of the operation justifies the expenditure, some of this fine pyrolusite can be recovered by reclaiming the heavier portion of the log-washer overflows in thickeners or settling ponds and re-treating this material by flotation. Samples from a settling pond at the Cedar Creek mill were tested by flotation. They were taken at the edge of the pond and are undoubtedly enriched by selective settling, nevertheless the test indicates the floatability of the mineral. The sample - assaying 25.3 percent manganese, 23.7 percent iron, 13.8 percent insoluble residues, and 0.251 percent phosphorus - was treated by flotation, and fair recovery of concentrates was made with a tenor of 42.9 percent manganese, 22.5 percent iron, and 2.5 percent insoluble residues. The type of reagents and the amounts used were similar to those shown in table 31.

TABLE 31. - Flotation Test of Composite Log-Washer Concentrates

Product	Weight, percent	Assay, percent		Percent of total	
		Mn	SiO ₂	Mn	SiO ₂
Cleaner concentrate.....	63.0	32.16	13.17	72.75	36.95
Cleaner tailings.....	3.9	14.41	39.45	2.00	7.98
Rougher concentrate.....	66.9	31.17	15.08	74.75	44.95
Rougher tailings.....	33.1	21.30	37.35	25.25	55.07
Composite.....	100.0	27.90	22.45	100.00	100.00

Pounds per ton ore			
Reagents used	Rougher	Cleaner	Total
Pine oil.....	0.06	0.06	0.12
Oleic acid.....	.18	.09	.27
Sodium silicate.....	2.00	1.50	3.50
Sodium carbonate.....	1.00	.50	1.50
Sodium oleate.....	.05	---	.05

Blue Ridge Region, Va.

Location and Extent

Virginia, the first producer of manganese ore, has numerous deposits in a belt paralleling the Appalachians and extending from the northern to the southwestern part of the State.

Before the World War Virginia produced 45 percent of the total manganese ore mined in this country. The greater portion of the 270,348 tons of ore produced in Virginia from 1867 to 1917 came from the Crimora mine. Since 1917 Virginia has produced, according to Mineral Resources of the United States, up to and including 1929, 48,436 tons of manganese ore, 23,638 tons of ferruginous manganese ore, and 5,094 tons of manganiferous iron ore (360).

The manganese deposits have been examined by geologists of the United States Geological Survey and the Virginia Geological Survey; and two excellent reports (358,360) have been published since 1918, in which the geology and the various deposits are described.

Most manganese ore of this district is found as residual deposits, resulting from the decay of the Shady dolomite. About 82 properties of this nature were described by Stose and his associated in their report (360).

A discussion of only some ores found in the Blue Ridge region will be given here. The deposits of the Oriskany formation, Woodstock district, will be discussed later.

Mineralogy

The ore is usually found as masses of oxides disseminated through clay. The predominant manganese mineral is psilomelane, with smaller amounts of manganite, pyrolusite, and wad. The psilomelane usually is in nodular form; much of it has a dull luster, indicating a high gangue content. Some of the manganese mineral also occurs as flat, shaly particles of light weight and low grade. Associated with the clay in some ores are particles of chert and shale.

Concentration Tests

Exploratory work during recent years has indicated the presence of ore at the Old Dominion property. Inasmuch as this deposit adjoins the Crimora mine, which produced approximately 160,000 tons of high-grade ore, samples from the Old Dominion mine were given particular attention.

Gravity Concentration. - A composite sample from drill holes on the Old Dominion property was examined. Much manganese was in the form of psilomelane nodules, although some of it occurred as flat shaly appearing particles. The clay had been washed out of the sample, so that the product examined may be said to correspond to a log-washer concentrate. The gangue consisted largely of altered chert with some shale. Much psilomelane had a dull luster and apparently contained considerable silica. This was brought out by a detailed examination and will be discussed in some detail later. A screen analysis, given in table 32, shows that the manganese in this ore, as in other residual ores, has a decided tendency to segregate in the coarser sizes.

Float-and-sink tests of fine sizes indicated that unless many of the lower-grade manganese-mineral particles could be rejected in the tailing gravity-concentration methods would yield siliceous concentrates and lower recovery. The results of these tests appear in table 33.

The experiments indicated that the manganese mineral contained much gangue so finely disseminated that it was not detectable with the naked eye or the low-power microscope. The percentages of gangue in the seemingly clean manganese particles varied widely. To determine the percentage of clean mineral and the relation between the specific gravity and gangue content of the mineral grains, the density of each particle of a sample of plus 3-mesh material was determined with a Joly balance. The free gangue and the visibly locked grains were removed before the test was begun. After the specific-gravity determinations the particles were combined into respective specific-gravity increments and analyzed. The results are shown in table 34.

TABLE 32. - Screen Analysis of Old Dominion Ore

	Weight, percent	Assay, percent			Percent of total		
		Mn	Fe	Insol.	Mn	Fe	Insol.
Plus 4 .	18.5	30.5	6.3	33.8	23.9	18.1	14.1
4 to 14	53.2	26.0	6.7	38.9	58.6	55.0	46.8
14 to 100	25.5	15.8	6.0	59.5	17.1	23.7	34.4
Minus 100 .	2.8	3.6	7.5	63.6	.4	3.2	4.7
Composite	100.0	23.6	6.5	44.2	100.0	100.0	100.0

TABLE 33. - Float-and-Sink Tests of Old Dominion Ore

Sieve	Product	Weight, percent	Assay, percent			Percent of total		
			Mn	Fe	Insol.	Mn	Fe	Insol.
4 to 14..	(Float on 2.92	30.9	5.9	3.5	80.6	6.9	16.4	64.1
	(Sink in 2.92	69.1	34.9	8.2	20.3	93.1	83.6	35.9
	Total	100.0	26.0	6.7	38.9	100.0	100.0	100.0
20 to 35.	(Float on 2.92	43.5	4.1	2.6	86.8	8.9	16.0	76.4
	(2.92 to 3.2	9.6	20.8	9.3	42.7	10.0	13.0	8.3
	(3.2 to 3.5	13.1	21.7	15.9	29.0	14.3	30.4	7.6
	(Sink in 3.5	33.8	39.4	8.4	11.3	66.8	40.6	7.7
	Total	100.0	19.9	6.9	49.4	100.0	100.0	100.0

TABLE 34. - Specific-Gravity Analysis of Manganiferous Ore
from Old Dominion Mine

Specific gravity	Weight, percent	Assay, percent			Percent of total		
		Mn	Fe	Insol.	Mn	Fe	Insol.
2.9 to 3.0	1.4	28.1	2.5	41.0	1.0	1.0	2.9
3.0 to 3.1	6.6	33.3	2.5	32.2	5.4	4.4	10.6
3.1 to 3.2	8.1	35.3	1.8	29.4	7.1	4.0	11.9
3.2 to 3.3	4.7	33.7	2.3	31.2	3.9	2.9	7.3
3.3 to 3.4	14.0	35.6	5.9	25.1	12.3	22.2	17.5
3.4 to 3.5	6.9	42.7	4.0	15.4	7.3	7.3	5.3
3.5 to 3.6	7.5	41.5	2.7	18.9	7.7	5.4	7.1
3.6 to 3.7	5.4	41.8	3.2	17.3	5.6	4.7	4.7
3.7 to 3.8	34.6	43.9	2.9	17.0	37.5	26.7	29.3
3.8 to 3.9	3.3	44.9	5.3	10.0	3.7	4.6	1.6
3.9 to 4.0	6.6	44.9	9.1	5.3	7.3	16.1	1.7
4.0 to 4.1	.9	54.0	3.0	1.4	1.2	.7	.1
Total ..	100.0	40.5	3.7	20.0	100.0	100.0	100.0

TABLE 35. - Float-and-Sink Test of $\frac{1}{2}$ -Inch to 10-Mesh
Old Dominion Ore

Product	Weight, percent	Assay, percent			Percent of total		
		Mn	Fe	Insol.	Mn	Fe	Insol.
Float on 2.95	29.8	4.6	2.6	85.4	4.7	14.3	72.3
Sink in 2.95	70.2	39.8	6.6	13.9	95.3	85.7	27.7
Total	100.0	29.3	5.4	35.2	100.0	100.0	100.0

TABLE 36. - Float-and-Sink Test of 20- to 35-Mesh
Page County Ore

Product	Weight, percent	Assay, percent			Percent of total		
		Mn	Fe	Insol.	Mn	Fe	Insol.
Float on 2.95	8.9	1.6	3.2	89.2	0.4	8.0	31.4
2.95 to 3.2	4.9	20.1	5.9	43.6	2.7	8.0	8.2
3.2 to 3.5	12.5	30.4	4.2	30.0	10.5	15.0	14.5
Sink in 3.5	73.7	42.5	3.3	16.3	86.4	69.0	45.9
Total	100.0	36.2	3.5	26.1	100.0	100.0	100.0

TABLE 37. - Flotation Test of Old Dominion Ore

Product	Weight, percent	Assay, percent		Percent of total	
		Mn	SiO ₂	Mn	SiO ₂
Cleaner concentrates	47.8	35.62	12.65	80.00	14.80
Cleaner tailings (middlings)	6.9	11.62	51.00	3.70	8.60
Tailings	45.3	7.64	69.05	16.30	76.60
Composite	100.0	21.28	40.85	100.00	100.00

Reagents used	Pounds per ton		
	Cleaner	Rougher	Total
Oleic acid	0.12	0.06	0.18
Sodium oleate ..	.24	.06	.30
Pine oil12	-	.12
Sodium silicate.	2.00	1.00	3.00
Sodium carbonate.	1.00	.50	1.50

Many of the particles of low specific gravity were flat and shaly. The grains of better grade were of a nodular shape, but except the heaviest, they also contained too much silica.

Some of the ore from the Old Dominion property does not contain so much shaly manganese material, as shown by examination of another sample that is more favorably inclined to concentration. A float-and-sink test of it is shown in table 35.

The presence of much silica in apparently clean psilomelane grains is common in the residual deposits of the Appalachians. Ores of similar character are found in other Virginia deposits and in Tennessee, as has been reported previously.

A float-and-sink test of an ore from Page County, Va. (table 36) shows the large amount of gangue contained in nodules of psilomelane, which to the naked eye are free of gangue. Apparently this condition is due to partial replacement of siliceous gangue mineral by manganese oxides. Grinding to extremely fine size is necessary to unlock the manganese minerals.

Attention is called to the fact that the assays for insoluble residues, which include both silica and alumina, have been shown in the preceding tests. These ores contain much alumina, hence the silica analyses are generally several points lower than the insoluble analyses. Cleaner ores than those just cited are believed to be present in the Blue Ridge region, however, it is apparent that many of the ores are made up largely of low-grade manganese minerals that will not yield concentrates containing less than about 12 percent silica. Such concentrates, while not attractive to producers of ferromanganese at present, would, however, find a ready market for blast-furnace use and could also be utilized for the production of a 70 percent ferromanganese during emergencies.

Flotation. - Several flotation tests of Old Dominion ore gave fair results. Grinding to 150-mesh did not give complete liberation, and the flotation concentrates had a tenor of about 13 percent of silica. The results of a typical test and the reagents used are given in table 37, and indicate a recovery of 80 percent of the manganese. This ore, however, floated quite slowly, and the advisability of applying flotation to ores of this type is still open to question.

Woodstock District, Va. (358)

Location and Extent

Because of the development of a large ore body and the erection of a modern mill at the Mineral Ridge or Cedar Creek property the Woodstock district requires attention. The ore formation is also somewhat different from the residual manganese deposits commonly found in the Appalachian region.

Among the manganese properties in the Woodstock district the Cedar Creek is outstanding. This property is on Manganese Ridge 11 miles north of Woodstock, Va., and has an interesting history. Manganese ore was first produced there in

1834, and the mine has been operated intermittently ever since. The production of ore in the earlier days was small, and the methods of mining and preparation were crude. Hand sorting and log washing were practiced. Estimates place the total production until April 1921 as 4,000 tons. The property was acquired in 1920 by the Hy-Grade Manganese Production & Sales Corporation, which has since carried out an exploration program. Production until 1930 came largely from the development work. Some of the ore was hand-picked and the rest treated in a small jigging and tabling plant. The value of the output up to 1928, according to C. W. Massie, manager of the property, was \$300,000. By 1929 exploration had proved the existence of a large ore body. A detailed geological survey of the property was made in 1920 by Stose and Miser.

Mineralogy

Pyrolusite and psilomelane are the most important manganese minerals, although some manganite and wad are also present. At some places pyrolusite accounts for most of the manganese, while at others psilomelane predominates. The pyrolusite is high grade, and much has been hand-picked and shipped during previous years for chemical ore. It is usually fine-grained and crumbles to a fine powder when handled. Psilomelane in the ore usually is found as nodules with a botryoidal surface and contains an appreciable amount of barium oxide. Manganite often is found as radiating crystals associated with the pyrolusite and psilomelane. Very little wad was seen in the samples examined, but according to Stose and Miser some of it occurs disseminated through pyrolusite and is said to contain 40 to 45 percent manganese. Associated with the manganese minerals are quantities of sandstone, chert, quartz, and clay and varying amounts of limonite. The phosphorus content of the ore shipped from this property has been 0.20 to 0.25 percent.

Concentration Tests

A study of the ore from the Cedar Creek mine indicates that a concentration process would consist of two steps, (1) washing the clay from the mixture of manganese oxides and chert and (2) separating this chert from the economic minerals. Because much of the manganese is in the form of soft, powdery pyrolusite which slimes badly, precautions must be taken to recover as much of the fine mineral as possible. Separation of the coarse minerals from the gangue is not particularly difficult. The ore is not badly locked, and crushing to one-half inch is sufficient. The predominant gangue mineral other than clay is a weathered chert of quite light weight; tests showed that it had a specific gravity of 2.15 to 2.62. In general, the specific gravity of the manganese minerals is more than 4, so that there is ample difference between the weights of the manganese and gangue to permit separation of the two with jigs and tables.

Cedar Creek Mill

Construction of a modern mill, with a rated capacity of 30,000 tons of ferro-grade ore per year, was begun in 1929 by the Hy-Grade Manganese Production & Sales Corporation and completed late in January 1930. Inasmuch as this mill is the newest in the country and embodies modern methods of concentration the flow sheet (fig. 7) will be discussed in some detail.

Because the general character of the ore treated is much the same as that in residual manganese deposits throughout the Appalachians, the same general flow sheet, with small changes to take care of the peculiarities of each ore, will no doubt be found applicable to other ores.

The ore is trammed to the mill grizzly, which has openings of about 4.5 inches. The oversize is low-grade and is rejected, and the undersize constitutes the mill feed.

The mill feed passes to a trommel with $1\frac{1}{4}$ -inch holes, where the screening is done without the addition of water. The undersize is washed in an Elmore washer to remove the clay. The washed material passes to a set of rolls, where it is crushed to one-half inch. The trommel oversize is crushed separately in a gyratory crusher.

The minus one-half inch feed is screened into 4 sizes by Hummer screens, and each of the 3 coarser sizes is treated separately on Harz-type jigs. The finest size and the undersize from the Elmore washer are deslimed in a Dorr classifier. The flow sheet shows the sand from the Dorr classifier distributed to three tables. A change has since been made so that the sand product passes to a hydraulic classifier, and a classified feed is fed to the tables.

Middlings from the jig are ground in a Hardinge ball mill and pumped back to the Elmore washer for re-treatment. The hutch product from the Harz jigs, plus the table middlings, are re-treated in a James jig. Slimes from the Dorr classifier and the hydraulic classifier are dewatered by three cones and concentrated on two slime tables.

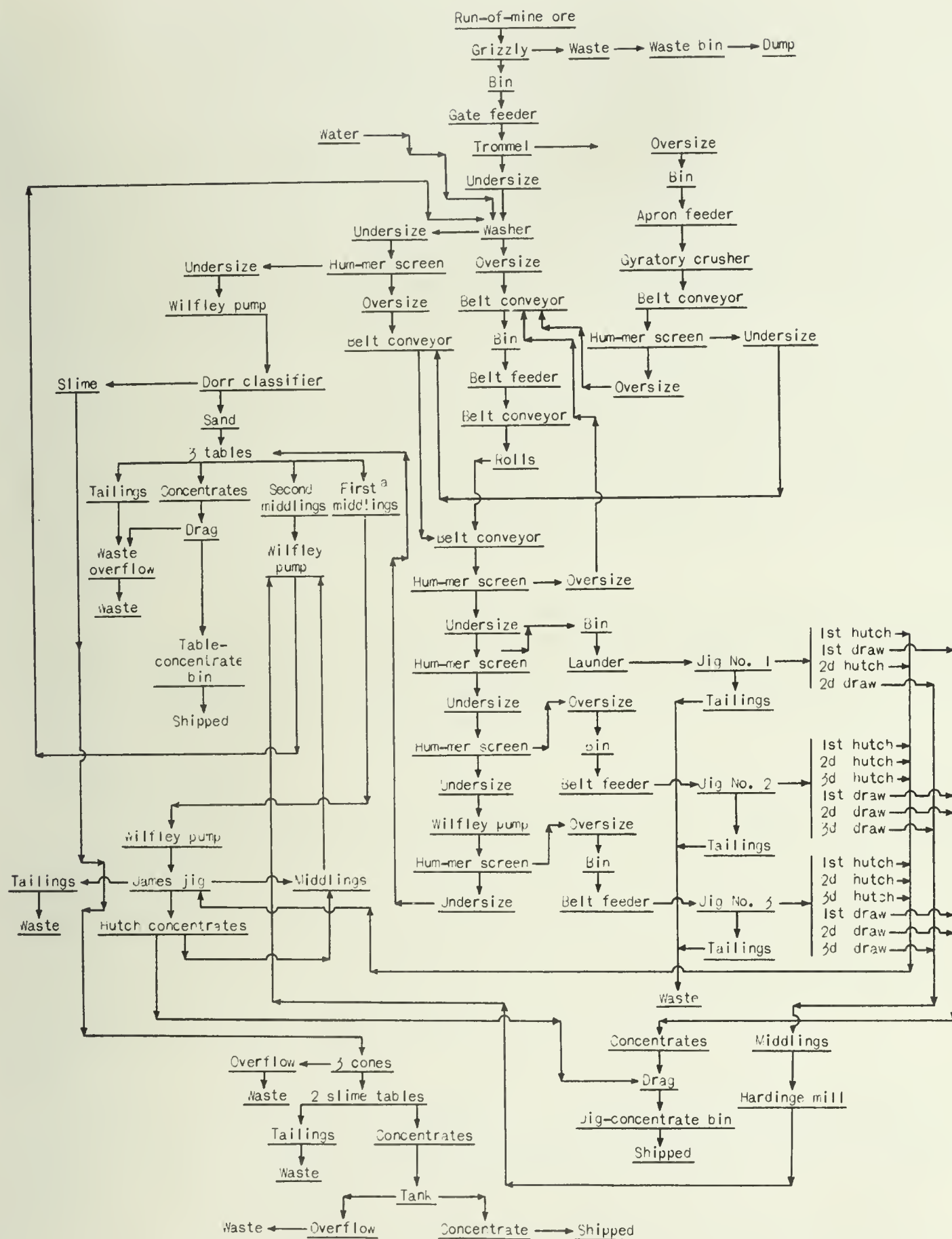
The advantage of feeding a classified product to tables instead of a distributed feed was brought out well at this mill. Installation of a multiple-spigot classifier resulted in an increase in the capacity of the tables and a decrease in the manganese content of the table tailings from about 11 to about 2 percent compared to results with a distributed feed.

The metallurgical results at this mill are quite satisfactory; the concentrates are suitable for metallurgical purposes, and the recovery of manganese is high considering the type of ore treated. It might be added that the company also hand-picks and cobs some of the ore in the open pit to produce a chemical grade product. The output of the mill, except a small tonnage ground by the company for use in the ceramic industry, is consumed by steel plants:

Olympic Peninsula, Wash.

Location and Extent

For many years, manganese-bearing ores have been known to occur in the Olympic peninsula; but little attention was given to them until the discovery in 1923 of the Crescent mine, which has produced high-grade metallurgical ore. Since that time prospecting has been done, and outcrops of manganese deposits have been discovered; however, none contain as high-grade ore as that produced by the Crescent mine.



^a First middlings nearest concentrate.

Figure 7.—Flow sheet of Hy-Grade Manganese Production and Sales Corporation, Cedar Creek Mill, Woodstock, Va.

The manganese deposits occur in a belt extending around the south, east, and north sides of the Olympic Mountains from near Humptulips to Beaver Creek, west of Lake Crescent, or 110 miles. The Humptulips deposits are at the southwestern end of this belt. Between these deposits and the South Fork of Skokomish River is a 20-mile stretch where manganiiferous float has been found, but the outcrops have not been located. A number of mines and prospects are located between the Skokomish River and Mount Constance, a distance of 25 miles. In the 20 miles between Mount Constance and Lake Crescent a few manganiiferous outcrops have been discovered. The area between Lake Crescent westward to Beaver Creek (25 miles) is marked by well-distributed outcrops, including that of the Crescent mine. The geology of the district has been described in detail by Pardee (289). The mineralogy of the various ore deposits will be discussed under "Concentration tests."

High-grade ore was discovered a mile west of Lake Crescent in 1923; and production began from this property, known as the Crescent mine, in 1924. The ore body was tabular (6 to 14 feet thick) and almost vertical. It had a maximum pitch length of 180 feet and a maximum slope length of 120 feet. During 1924, 1925, and 1926 this ore body produced 16,275 long tons of high-grade manganese ore and is now thought to be worked out. Efforts of the operators to find other high-grade ores at this mine have not been successful, although considerable exploration has been done.

The ore at this mine consisted of a hard, dense mixture made up principally of hausmannite, with some bementite and a manganiiferous carbonate. It was of excellent quality for metallurgical purposes, as is shown by the following analysis of shipments made in 1924, aggregating about 5,000 tons: Manganese, 52.88 percent; iron, 0.83 percent; silica, 8.80 percent; and phosphorus, 0.05 percent.

Concentration Tests

Much of the manganiiferous material in the Olympic Peninsula consists largely of the manganese silicates, bementite predominating, with minor amounts of manganese oxides, hematite, and quartz. Small deposits of high-grade ore similar to that at the Crescent mine may be found, but it is evident that by far the greatest part of the manganese is in the form of siliceous material. Obviously, ores in which the manganese is chemically combined with silica cannot be concentrated by ore-dressing methods to produce low-silica concentrates. However, many samples of manganiiferous ores from outcrops of the district were examined to ascertain their composition and character and to determine whether any of them could be cleaned by ore-dressing methods.

Samples representing the manganiiferous deposits of the district were obtained from three general localities: (1) Crescent area (Lake Crescent to Beaver Creek); (2) Constance area (approximately 13 miles west of Brinnon); and (3) Humptulips area (Humptulips River Basin, 30 miles north of Hoquiam).

Except for the sample taken from the Crescent tunnel all were from surface trenches, no underground workings having been opened. Inasmuch as weathering has not altered the deposits more than a foot or so below the surface it is believed that the manganiiferous deposits will not vary greatly at depth from the samples obtained by trenching. The analyses of the samples and the approximate locality from which they came are given in table 38.

TABLE 38. - Locations and Analyses of Manganiferous Material
from Olympic Peninsula

District	Assay, percent				Description of sample
	Mn	Fe	Insol.	P	
Crescent....	35.9	1.6	22.5	0.041	General sample from 3 surface showings
	48.3	.4	15.5	.053	Selected high-grade specimens
	44.8	1.7	16.1	--	State lease claim
	26.2	3.2	21.9	--	Low-grade ore from Crescent tunnel
Constance...	31.2	17.1	16.3	.079	Elkhorn claims
	45.4	4.9	22.9	.083	Achilles property
	10.7	23.9	37.4	--	3 showings, east lead claim No. 4
	26.7	9.9	37.1	--	Middle lead claim No. 6
	20.8	21.6	22.9	--	West lead claim No. 15
	28.4	18.9	21.8	--	General sample
Humtulpis..	14.4	21.6	40.9	.062	General sample of outcrop ore
	36.6	2.0	27.0	.050	Selected high-grade specimens
	11.4	32.7	24.6	--	Hand specimens of typical ore

The material from all three districts is similar. The manganese occurs as a silicate, and the phosphorus content is uniformly low. The ore from the Crescent district is characterized by its relatively high manganese and low iron contents. Ore from the Constance area is somewhat lower in manganese and higher in iron. The Humtulpis ore usually contains considerable hematite and is relatively low in manganese.

Crescent Area. - Examination of the general sample from this district showed that it was made up of a fine-grained aggregate of hausmannite and bementite and a complex manganese carbonate. Little free gangue was present, and float-and-sink tests showed virtually no separation. A sample from the State lease claim was the only one which responded to gravity concentration. This sample was hausmannite and a relatively light-weight manganese silicate. Float-and-sink tests of this material gave a concentrate containing only 4.8 percent insoluble. The results of these tests, with those of the Crescent tunnel sample (which proved to be similar to the general sample, except that it contained more bementite and manganese carbonate), are given in table 39.

No separation could be made of any of the three samples by magnetic methods, virtually the entire sample in each case being attracted by a high-intensity magnet.

Constance Area. - Most of the ore from this district was hard and brittle and broke with a conchoidal fracture. Bementite was the predominant mineral, but a considerable amount of dense reddish mineral with a vitreous luster and a specific gravity of 3.1 was also present. This was assumed to be neotocite.

TABLE 39. - Float-and-Sink Concentration of 20-to 35-Mesh Olympic Peninsula Manganese Ores
with Heavy Liquid of 2.95 Specific Gravity

Sample	Feed, percent		Concentrates, percent				Tailings, percent			Recovered Mn, percent	Rejected Fe, percent	Rejected insol., percent
	Mn	Fe	Insol.	Weight, percent	Mn	Fe	Insol.	Mn	Fe	Insol.		
Crescent area:												
General sample ...	38.1	2.7	22.5	93.8	39.2	2.6	22.0	21.0	3.6	29.1	8.3	8.0
State lease	44.3	1.4	15.9	65.5	53.3	1.0	4.8	27.1	2.2	37.0	53.5	80.2
Crescent tunnel ..	25.9	4.7	20.6	85.7	27.9	3.6	20.7	13.9	4.0	20.4	12.1	14.2
Constance area:												
Elkhorn	-	-	15.3	98.8	30.0	19.3	15.2	-	-	22.9	-	-
Achilles	45.0	-	23.3	97.1	45.5	2.7	23.1	28.8	-	29.3	-	-
East lead	11.1	24.2	36.2	84.9	11.4	27.0	30.3	9.9	8.4	69.4	5.3	28.9
Middle lead	27.0	10.7	37.4	64.2	38.2	13.6	15.0	7.1	5.7	76.6	19.1	74.1
West lead	20.8	21.5	23.0	92.5	20.4	22.6	22.3	26.0	7.5	31.2	2.6	10.1
General sample ...	28.2	19.9	21.2	91.5	29.1	21.3	18.5	19.0	6.2	50.5	2.6	20.3
Humtulsips area:												
General sample ...	14.6	20.8	40.5	62.7	17.2	26.7	28.4	10.3	10.9	60.9	19.6	56.1
Hand specimens ...	11.3	32.7	23.1	79.8	12.2	34.2	20.5	7.6	26.1	33.3	16.2	29.1

Little free gangue could be observed, and the presence of iron in the oxide form could not be detected by the eye. In many ways these ores resemble a slag both in appearance and in the manner in which the mineral constituents occur. Float-and-sink tests of samples of these ores, sized between 20- and 35-mesh, were very unpromising, only one ore giving a concentrate as low as 15 percent insoluble (see table 39). Two of the samples contained material that was strongly magnetic and could be separated with a hand magnet. The remainder of the material was attracted by a high-intensity magnet. The strongly magnetic portions were the higher in iron, and it is probable that they contained some magnetite particles of minute size, although none were detected.

Humptulips Area. - The samples from this area contained considerable hematite and appeared to have a much higher manganese content than the chemical analyses showed. The illusion is due to the formation of superficial oxide coatings along fracture planes; and as the ore breaks along these fractures many of the particles appear to be solid manganese oxides, while in reality all but the outer film of high-grade oxide consists of manganese silicates.

Float-and-sink tests of two samples showed that some enrichment could be made by gravity concentration but that in all probability the concentrates would contain not less than 20 percent silica. The results are shown in table 39.

Approximately 10 percent of the composite sample was strongly magnetic, 84 percent was weakly magnetic, and the remaining 6 percent was nonmagnetic. The strongly magnetic material is higher in manganese and lower in iron than the weakly magnetic portion. This strongly magnetic condition can only be explained by assuming that the higher-grade manganese oxides are associated with a small amount of magnetite. The iron mineral in the weakly magnetic portion is undoubtedly hematite. The results of magnetic tests are shown in table 40.

TABLE 40. - Magnetic Concentration of Some Olympic Manganese Ores, 20- to 35-mesh

Sample	Magnetic intensity ¹	Weight, percent	Assay, percent			Percent of total		
			Mn	Fe	Insol.	Mn	Fe	Insol.
Constance (middle lead)	Strongly magnetic..	23.4	33.1	16.9	12.1	26.5	31.3	13.6
	Weakly magnetic....	74.8	28.7	11.6	22.2	73.5	68.7	79.9
	Nonmagnetic.....	1.8	---	---	74.4	---	---	6.5
	Total.....	100.0	29.2	12.6	20.8	100.0	100.0	100.0
Constance (general sample)	Strongly magnetic..	32.3	25.0	23.0	20.1	28.5	45.3	28.1
	Weakly magnetic....	67.7	29.9	13.2	24.5	71.5	54.7	71.9
	Total.....	100.0	28.3	16.4	23.1	100.0	100.0	100.0
Humptulips (general sample)	Strongly magnetic..	10.4	30.8	14.1	15.4	15.8	8.2	5.5
	Weakly magnetic....	83.8	20.3	19.1	27.0	83.9	88.9	78.4
	Nonmagnetic.....	5.8	1.0	9.1	80.2	.3	2.9	16.1
	Total.....	100.0	18.6	18.4	30.5	100.0	100.0	100.0

¹ The strongly magnetic material was removed with a hand magnet, while the weakly magnetic material was removed with a high-intensity magnet.

SUMMARY AND CONCLUSIONS

The supply of domestic manganese ore having over 35 percent of manganese in the crude state is not great, but the present investigation indicates that the tonnage of ore that can be concentrated to this grade is large.

In the course of an investigation by the United States Bureau of Mines 116 ores from 20 States were examined to determine their amenability to concentration. A description of 17 of the more important manganese deposits, with experimental work on ores from them, is reported in some detail. These individual reports show the character of the ores and give results indicative of what may be expected by concentration. Of the 116 ores examined, 20 may be concentrated to meet all requirements for ore suitable for the manufacture of standard 80-percent ferromanganese, and 43 were found of suitable composition for use in the production of spiegeleisen or high-manganese pig iron.

During the investigation a flotation method for the treatment of both carbonate and oxide ores was developed. This process depends on selective flotation of the manganese oxide and carbonate minerals away from the gangue by the use of fatty acids or fatty acid soaps and suitable conditioning agents, such as sodium silicate, and a frothing agent. For most ores an alkaline circuit is necessary, but in some instances the use of a slightly acid circuit in the cleaning stages is desirable. This process is especially applicable to the treatment of the low-grade carbonate ores of the Butte district. The Anaconda Copper Mining Co. has demonstrated successfully the practicability of the flotation of these ores and has produced low-phosphorus concentrates which when sintered have 60 to 62 percent manganese and about 7 percent silica. The cost of milling ores by this process, according to company officials, is \$1.00 to \$1.50 per ton, and the by-product value of the metallics recovered incident to the flotation of the manganese will compensate for about half of this cost.

Particular attention has been given the ferruginous manganese ores of the Cuyuna district, Minn. The close association of the iron and manganese minerals prevents the production of ferro-grade ores from such material by existing ore-dressing methods. However, much of the silica can be eliminated by these methods, making the ore suitable for other metallurgical purposes. A 30-percent spiegeleisen is probably the highest alloy of manganese that can be manufactured directly from the concentrates.

In the summer of 1930 a 10-table pilot plant with a supplementary flotation unit was built and operated at Crosby, Minn. It demonstrated that certain manganeseiferous ores of that district could be concentrated. During the summer of 1931 this plant produced approximately 25,000 tons of concentrate containing less than 10 percent silica from ore formerly regarded as untreatable.

Due to the physical and chemical nature of the ores, the concentration of the large low-grade deposits of the Chamberlain (S. Dak.) and the Kingman (Ariz.) districts by standard ore-dressing methods does not appear promising. The use of such ores depends on the successful development of leaching or pyrometallurgical processes.

The manganiferous deposits of the Olympic Peninsula of Washington are largely manganese silicates. Ore-dressing methods in a few instances can be used to reject some of the gangue, but in general such methods fail. It is possible that some of the ores from the district may be used for the manufacture of silicomanganese or silicospiegel without concentration. Such methods have not, however, been developed.

The supplies of ferruginous manganese ores, manganiferous iron ores, and zinc residues from the plants of the New Jersey Zinc Co. are believed to be entirely adequate to fill the requirements for the production of spiegeleisen and manganiferous pig iron. The adequacy of the reserves of ore of this type have never been seriously questioned.

As in the past, Georgia, Virginia, and New Mexico will continue to produce manganese ore. The amount will depend on the demand.

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MANGANESE; ITS OCCURRENCE, MILLING, AND METALLURGY

PART II.

CHAPTER 5. THERMODYNAMIC PROPERTIES OF MANGANESE, AND
ITS METALLURGICALLY IMPORTANT COMPOUNDS



BY

C. G. MAIER

INFORMATION CIRCULAR

UNITED STATES BUREAU OF MINES

MANGANESE; ITS OCCURRENCE, MILLING,
AND METALLURGY¹

Part II

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CHAPTER 5. THERMODYNAMIC PROPERTIES OF MANGANESE,
AND ITS METALLURGICALLY IMPORTANT COMPOUNDS

By C. G. Maier²

The present trend of chemicometallurgical research toward widespread and inclusive use of thermodynamic methods requires the inclusion of available data of this sort in any comprehensive metallurgical report. Such data for manganese are far from adequate for the full utilization of these methods. Even if our knowledge of these properties is weak, the collection of available figures is nevertheless well worth while, especially if their utilization in an approximate way may show inherent possibilities of application. No less important is the necessity for discovering the vital gaps in the existing information, to serve as a basis for further pertinent study.

The utility of the thermodynamic methods of Lewis and Randall (234)³ has become increasingly evident to metallurgists. It is true that at present no better approach to the general, fundamental problems of chemical metallurgy is available than close adherence to these methods. The effort is here made to collect and correlate, as far as possible, significant quantitative thermodynamic data on manganese and its metallurgically important compounds. There are considered here only the intrinsic properties of pure materials. The order maintained as far as possible will be as follows:

1. Specific heats at low temperatures.
 - (a) Compressibility, thermal expansiveness and atomic volume.
 - (b) Entropy calculated from specific heats.
2. Specific heats at high temperatures.
3. Sublimation and vapor pressures.
4. Thermal data.
 - (a) Heats involved in change of state; sublimation, fusion, vaporization, transitions.
 - (b) Heats of formation of compounds.
 - (c) Free energies of formation.
5. Correlation of heats, free energies, and entropies, where possible.

METALLIC MANGANESE

Allotropy

The question of the allotropy of manganese has already been discussed. There would seem to be little doubt as to the actual existence of at least three forms of manganese, but from a thermodynamic viewpoint interest is confined to transformations having definite and measurable energy quantities

² Supervising Engineer, fundamental studies section, Metallurgical Division, U. S. Bureau of Mines.

³ Throughout parts I to IV, numbers in parentheses refer to items in this bibliography, and page numbers to those in the reference cited.

associated with the change of form. Here recourse must be had to specific heat measurements at high temperatures. In a later section it will be shown that the available specific heat measurements indicate such energy changes at 835° and 1,044°C. only.

Comparison of these transition temperatures with those obtained by means other than specific heat measurements seems very unsatisfactory, but it must be remembered that such changes often show time and temperature lags, especially the latter when impure materials are considered. For the present purpose, therefore, it may suffice to accept the points derived from specific heats as valid, and the assumption will be made that up to 835°C the α form is stable, from 835 to 1,044° the β form exists, and above 1,044° the specific-heat methods really involved the tetragonal γ form.

Specific Heats at Low Temperatures

No measurements of true specific heat at low temperatures are reported. A few measurements of mean specific heat have been made by Dewar (70), by Estrieher and Staniewski (86), and by Nordmeyer and Bernulli (277). In a recent resume of low-temperature specific heat and entropy data, Kelley (200) has selected the data of Dewar as most suitable for entropy calculations, and reports $S_{298} = 7.3 \pm 0.5$.

Determination of the true specific heat of manganese metal at low temperatures is an assigned problem for investigation at the Pacific Experiment Station of the Bureau of Mines.

Compressibility, Expansion, and Atomic Volume

The only available determination of the compressibility of manganese is that of Richards (315), who reported $0.84 \cdot 10^{-6}$ per megabar and who also gave 7.7 for the atomic volume and 7.37 for the density at 20°C. Available figures for thermal expansiveness are those of Disch (71), who gives $1.59 \cdot 10^{-5}$ between -190 and 0°C., and $2.28 \cdot 10^{-5}$ between 0° and 100°C. The difference between the specific heats at constant pressure and volume may be calculated from the thermodynamic expression

$$C_p - C_v = \frac{\alpha^2 v T}{\beta} \quad (\text{Equation 1})$$

where C_p and C_v are the respective specific heats at constant pressure and volume, α the linear expansion coefficient, v the specific volume, T the absolute temperature, and β the compression coefficient. At 25°C. substitution of the various figures given results in the value 0.3 calorie per degree for the specific heat difference.

A recent determination of the density of manganese (44) indicates a value of 7.76 for the gram-atomic volume compared with the older figure, 7.34. The corresponding densities are 7.08 and 7.48. It seems, however, that Campbell

had for his density determinations electrolytic material of the ∇ or tetragonal form, and it is not strange that his results should be low.

According to Matsuyama (244), manganese contracts 1.7 percent of its volume during solidification. Assuming that the coefficient of expansion of the three forms of manganese can be considered equal for approximate calculation the densities and specific volumes may be summarized, as in table 41.

TABLE 41. - Densities and Specific Volume of Manganese

Form of Mn	Type measurement	Temperature, °C.	Density	Specific volume
α	Direct.....	20	7.37	7.45
α	X ray.....	20	7.44	7.38
α	Calc.....	835	7.04	7.80
β	X ray.....	20	7.29	7.54
β	Calc.(X ray)...	835	6.89	7.98
γ	Direct.....	20	7.08	7.76
γ	X ray.....	20	7.21	7.62
γ	Calc.(X ray)...	1,044	6.73	8.16
γ	do.....	1,220	6.65	8.26
Liquid.....	Calc.....	1,220	6.54	8.40

Specific Heats at High Temperatures

Available data on the specific heat of manganese at high temperatures consist of the measurements of Umino (373), of Wüst, Meuthen, and Durer, (401), of Stücker (363), and of Lämmel (226).

K. K. Kelley, of the Pacific Experiment Station of the Bureau of Mines, has recently made a critical study of high-temperature specific heats in connection with a bulletin on this subject now in preparation. He reports that the data of Umino are the best available, as determined by a comparative plot of total heats as determined by all these experimenters. Table 42 shows the total heats taken from Kelley's smoothed curve.

The reader will note that the melting point of manganese is taken at 1,220°C. Umino's figure was 1,221°C. Some nine other figures are given in the Landolt-Börnstein Tabellen (227), and Burgess (42) has reported a value obtained by a micromelting apparatus.

The mean of all values (11 in number) is $1,223^{\circ} \pm 13$, but of these, two sets, including that of Burgess, are discordantly high.⁴ When the two high values are omitted from the average it becomes $1,217^{\circ} \pm 8$. This figure has

⁴ Several values in the Tabellen are based upon a comparison which involved taking the melting point of nickel at 1,484°C. These figures have been corrected in the average by the author to correspond to a melting point for nickel of 1452.3°C.

been rounded off to 1,220° and has been used as more consistent with the thermodynamic data available than the higher figure of Gayler (108) given in chapter 1.

TABLE 42. - Total Heats for Manganese Metal from 0°C., Calories per Gram-Atom

Form of Mn	Temperature, °C.	Heat content (curve)	Difference, curve-Umino, percent
α	0	0	0
	50	299	---
	100	617	1.3
	150	953	---
	200	1,308	.5
	250	1,681	---
	300	2,072	1.0
	350	2,482	.7
	400	2,911	1.3
	450	3,358	.2
	500	3,823	-.2
	550	4,307	-.4
	600	4,809	-.2
	650	5,330	-.5
	700	5,869	-.3
	750	6,427	---
	800	7,003	---
	835	7,417	---
α - β	835	148	-6.0
β	835	7,565	---
	850	7,700	0.9
	900	8,150	.2
	950	8,600	-1.1
	1,000	9,050	-.9
β - γ	1,044	9,446	---
	1,044	244	-2.0
γ	1,044	9,690	---
	1,050	9,750	+.1
	1,100	10,250	---
	1,150	10,750	-1.0
	1,200	11,250	-1.0
	1,220	11,450	---
γ -liquid	1,220	3,576	+5
Liquid	1,220	15,026	---
	1,250	15,365	-.1
	1,300	15,930	+.4
	1,350	16,495	+.1
	1,400	17,060	-.1

For many purposes the figures in table 42 may be found more convenient than analytical expressions for specific heat, since some complications due to allotropy may be avoided in certain cases by the summation of total heats. Kelley has derived certain analytic expressions for heat content of the various forms, which are summarized in table 43.

TABLE 43. - Heat Content of Manganese From 0° C.
(Formula, $H_c = a + bt + ct^2$ ---)

Form of Mn	<u>a</u>	<u>b</u>	<u>c</u>	Temperature range, °C.
α	0	5.80	$3.692 \cdot 10^{-3}$	0 - 835
β	50.0	9.00	---	835 - 1,044
γ	-750	10.00	---	1,044-1,220
Liquid	1,240	11.30	---	1,220-1,400

By differentiation of these expressions and substitution of $T = t + 273.1$, the following equations are derivable for true specific heat.

$$C_p(\alpha \text{ Mn}) = 3.78 + 7.38 \cdot 10^{-3} T,$$

$$C_p(\beta \text{ Mn}) = 9.00,$$

$$C_p(\gamma \text{ Mn}) = 10.00,$$

$$C_p(\text{liq.}) = 11.30.$$

Sublimation and Vapor Pressures

Actual measurements of vapor pressures of manganese metal are meager. Greenwood (122, 123) reported a boiling point of 1,900°C. Observed temperatures varied about 100°; mean values were stated to be concordant to 20°. The melt was contained in a carbon crucible with MgO lining. In this connection, Tiede and Birnbrauer (372) have shown that manganese readily reduced MgO at higher temperatures. This would justify the assumption that the observed boiling point of Greenwood was too low.

Ruff and Bormann (328) determined the boiling point at 30 mm pressure for pure metal to be 1,511°C. Millar (258) has calculated vapor pressures for manganese, using Greenwood's boiling point combined with Wüst's specific-heat data.

In a recent article by Gayler (109) pure manganese is said to volatilize under 1 to 2 mm pressure at a temperature just above the melting point (1,225°C).

It seems highly probable that the measurements of Ruff and Bormann are the best available. If the assumption is made that these latter figures are approximately correct, a much more satisfactory calculation of the vapor pressure of manganese becomes possible, especially since the energy of the gas may be determined from spectroscopic data. Through the courtesy of Prof. W. F. Giaque (111) it is possible to include here recent results of such a calculation from spectroscopic data, the specific results being given in table 44.

TABLE 44. - Free Energy of Manganese Vapor from Spectroscopic Data

$T, ^\circ K.$	$-\frac{F^0 - E_0^0}{T}$	$T, ^\circ K.$	$-\frac{F^0 - E_0^0}{T}$
293.1	36.534	1,700	45.182
500	39.103	1,900	45.734
700	40.774	2,100	46.231
900	42.022	2,300	46.683
1,100	43.019	2,500	47.098
1,300	43.849	2,700	47.481
1,500	44.560	2,900	47.838

The symbol F^0 in table 44 refers to the free energy of the vapor in the standard state, and E_0^0 is the energy in the standard state at absolute zero. For further discussion of the methods used in obtaining these figures and explanation of the symbols the reader is referred to the article by Professor Giauque (110).

To use these figures for calculation of vapor pressures similar information is necessary with respect to the solid and liquid. Thus, assuming the solid to be the standard state, equation (30) in Giauque's paper above may be used:

$$F^0 - E_0^0 = \int_0^T \frac{T}{C_p} dt - T \int_0^T \frac{T}{C_p} d \ln T. \quad (\text{Equation 2})$$

The first integral is obviously the total heat from $0^\circ K.$ and the second the entropy. For the present purpose the specific heats for the various forms of manganese may be integrated more conveniently from $0^\circ C.$ or from $298.1^\circ K.$ in the case of total heat and of entropy, respectively, with the addition of appropriate corrections from low-temperature data to carry the results to absolute zero. Thus, in the case of entropy the addition involved here amounts to 7.3 units, as given previously (see p.100). Similarly, by drawing a Debye curve through and below Dewar's low-temperature data for manganese and extending it upward to meet the herein accepted value for the specific heat at $0^\circ C.$, as previously given, the total heat of manganese up to $273.1^\circ K.$ may be obtained as $1,000 \pm 50$ calories. The total heats at various upper temperatures are then given by the addition of 1,000 calories to the figures of table 42, and the corresponding entropies are calculated from the formulas given for specific heat, with suitable corrections for the transitions involved.

The results of such a series of calculations are given in table 45.

To calculate vapor pressures, consider the equilibrium $Mn_{(l)} = Mn_{(g)}$ at $1,511^\circ C.$ or $1,784^\circ K.$, where, according to Ruff and Bormann, the vapor pressure is 30 mm.

$$\text{Then, } \Delta F_{1,784}^{\circ} = -R T \ln \frac{30}{760},$$

$$\begin{aligned} \frac{\Delta F^{\circ}}{T} &= -R \cdot 2.3026 \log \frac{30}{760}, \\ &= 6.43. \end{aligned}$$

Now for the reaction, by interpolation in tables 44 and 45, for $T = 1,784$,

$$-\frac{(F^{\circ} - E_0^{\circ})}{T}(\text{gas}) - \frac{(F^{\circ} - E_0^{\circ})}{T}(1) = -\frac{\Delta F^{\circ}}{T} + \frac{\Delta E_0^{\circ}}{T},$$

$$45.42 - 15.50 = -6.43 + \frac{\Delta E_0^{\circ}}{T},$$

$$\frac{(\Delta E_0^{\circ})}{T}(1,784) = 36.35.$$

From this last result, corresponding figures for $\frac{\Delta E_0^{\circ}}{T}$ at any desired temperatures are calculated directly and combined with the values of $\frac{F^{\circ} - E_0^{\circ}}{T}$ for solid, liquid and gas to furnish directly the free energy of vaporization. Such calculated results are shown in table 46. Since $\frac{\Delta F^{\circ}}{T} = -R \ln p$, the corresponding vapor pressures are readily calculated and are shown in various forms in the last three columns of the table. For purposes of interpolation a curve showing the variation of $\log \text{V.P. v. } T^{\circ}\text{C.}$ is reproduced in figure 8.

TABLE 45. - Free Energy of Solid and Liquid Manganese

Form of Mn	Temperature		H	$\frac{H}{T}$	S	$-\frac{F^{\circ} - E_0^{\circ}}{T}$
	$^{\circ}\text{C.}$	$^{\circ}\text{K.}$				
α	25	298.1	1,146	3.84	7.3	3.46
	835	1,108	8,417	7.59	18.43	10.84
β	835	1,108	8,565	7.72	18.56	10.84
	900	1,173	9,150	7.80	19.07	11.27
	1,000	1,273	10,050	7.86	19.82	11.96
	1,044	1,317	10,446	7.92	20.49	12.57
γ	1,044	1,317	10,690	8.11	20.68	12.57
	1,100	1,373	11,250	8.19	21.08	12.89
	1,200	1,473	12,250	8.31	21.86	13.55
	1,220	1,493	12,450	8.34	21.92	13.58
Liquid....	1,220	1,493	16,026	10.73	24.31	13.58
	1,300	1,573	16,930	10.76	24.90	14.14
	1,400	1,673	18,060	10.80	25.60	14.80
	1,500	1,773	19,190	10.82	26.26	15.44
	1,600	1,873	20,320	10.85	26.88	16.03
	1,700	1,973	21,450	10.87	27.46	16.59
	1,800	2,073	22,580	10.89	28.02	17.13
	1,900	2,173	23,710	10.91	28.56	17.65
	2,000	2,273	24,840	10.93	29.07	18.14

In comparing the calculated vapor pressures in table 46 with the available direct determinations the reader will note that the boiling point is 2,000°C. compared to Greenwood's 1,900°, the difference being just that given by Greenwood himself as the reproducibility of his results. Furthermore, at the melting point the figure 1.46 mm is calculated, where the previously mentioned data of Gayler indicate a vapor pressure of 1 to 2 mm, an almost exact concordance but considerably different from Millar's calculated figures. The vapor pressure at 1,250°C., corresponding to Gayler's melting point, would be just a little over 2 mm.

Although application of the calculation of energy from spectroscopic data to vapor pressures still involves the supposition of monatomic gas in this case the figures so obtained are undoubtedly far superior to the usual calculation from the Sackur equation, which was essentially the basis of Millar's method. Should a better boiling-point or vapor-pressure measurement than that of Ruff and Bormann become available, recalculation from the original figures of table 44 would leave little to be desired, since the errors in the calculation must come almost entirely from the fixed point taken and from the free-energy figures for solid and liquid. Even with these limitations in mind the results must be regarded as highly satisfactory.

TABLE 46. - Calculated Vapor Pressures of Manganese Metal

Form of Mn	Temperature		$-\frac{F^0-E_0^0}{T}$		$\frac{\Delta F_0^0}{T}$, (s)-(g) or (l)-(g)	$\frac{\Delta F^0}{T}$	Log V.F.	V.P.	
			Gas	Solid or liquid				Atmospheres	Millimeters
	°C.	°K.							
α	25	298.1	36.53	3.46	217.25	184.18	-40.22	$6 \cdot 10^{-41}$	
	835	1,108	43.05	10.84	58.48	26.27	-5.740	$1.82 \cdot 10^{-6}$	$1.3 \cdot 10^{-3}$
β	835	1,108	43.05	10.84	58.48	26.27	-5.740	$1.82 \cdot 10^{-6}$	$1.3 \cdot 10^{-3}$
	900	1,173	45.33	11.27	55.25	23.19	-5.063	$8.65 \cdot 10^{-6}$	$6.58 \cdot 10^{-2}$
γ	1,000	1,273	43.74	11.96	50.90	19.12	-4.178	$6.64 \cdot 10^{-5}$	$5.05 \cdot 10^{-2}$
	1,044	1,317	43.91	12.57	49.20	17.86	-3.903	$1.25 \cdot 10^{-4}$.0950
	1,044	1,317	43.91	12.57	49.20	17.86	-3.903	$1.25 \cdot 10^{-4}$.0950
	1,100	1,373	44.12	12.89	47.20	15.97	-3.490	$3.24 \cdot 10^{-4}$.246
	1,200	1,473	44.47	13.55	44.00	13.08	-2.857	$1.39 \cdot 10^{-3}$	1.06
	1,220	1,493	44.54	13.58	43.41	12.45	-2.720	$1.915 \cdot 10^{-3}$	1.455
	1,220	1,493	44.54	13.58	43.41	12.45	-2.720	$1.915 \cdot 10^{-3}$	1.455
Liquid	1,300	1,573	44.79	14.14	41.21	10.56	-2.308	$4.92 \cdot 10^{-3}$	3.74
	1,400	1,673	45.10	14.80	38.74	8.44	-1.844	$1.43 \cdot 10^{-2}$	10.9
	1,500	1,773	45.39	15.44	36.58	6.63	-1.448	$3.57 \cdot 10^{-2}$	28.7
	1,600	1,873	45.66	16.03	34.61	4.98	-1.088	$8.17 \cdot 10^{-2}$	62.1
	1,700	1,973	45.92	16.59	32.87	3.54	-.773	$1.69 \cdot 10^{-1}$	128.5
	1,800	2,073	46.17	17.13	31.26	2.22	-.485	$3.27 \cdot 10^{-1}$	249.
	1,900	2,173	46.40	17.65	29.81	1.06	-.232	$5.86 \cdot 10^{-1}$	446.
	2,000	2,273	46.62	18.14	28.50	.02	-.004	$9.99 \cdot 10^{-1}$	760.

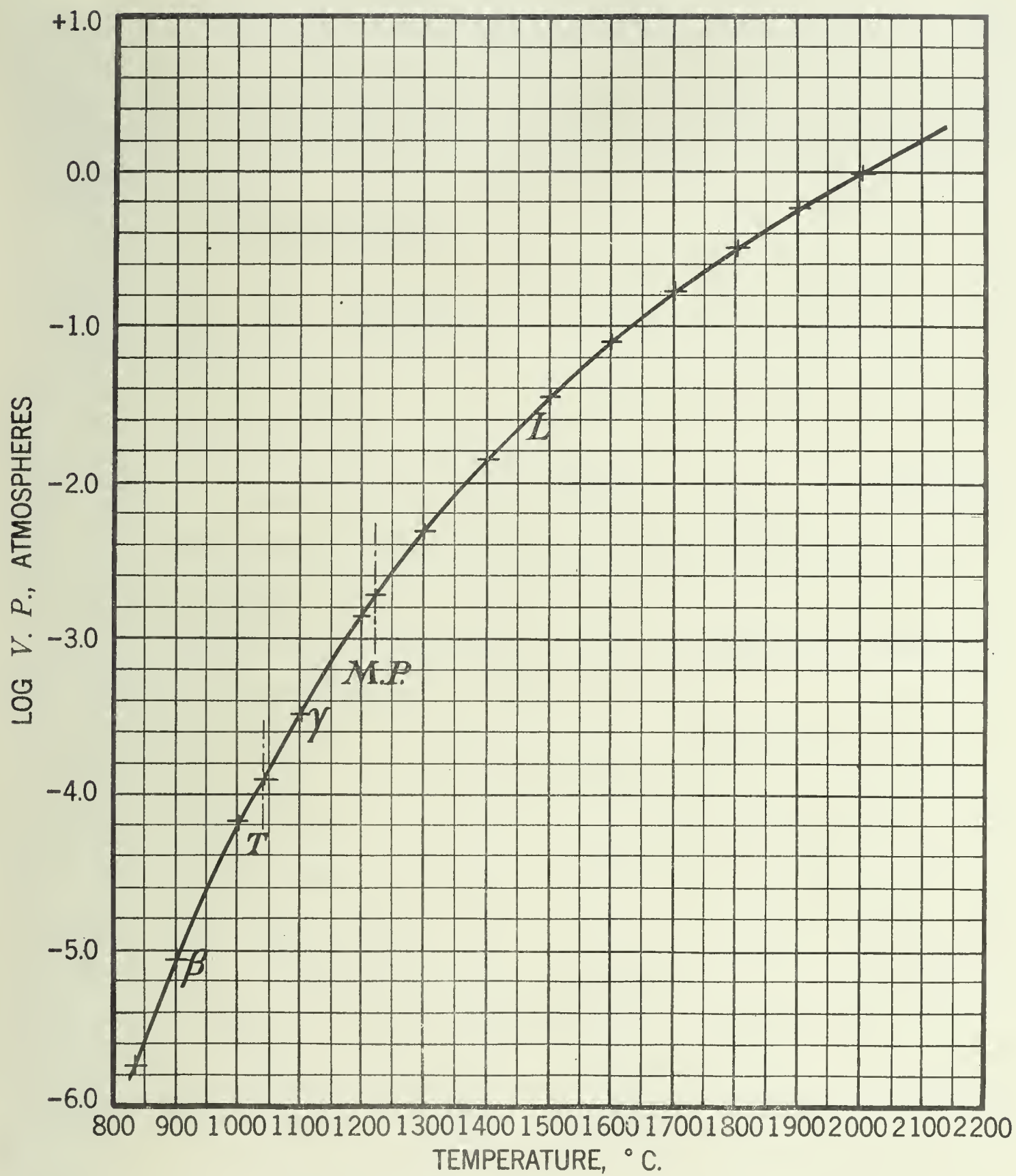


Figure 8.- Vapor pressure of manganese metal; L, Liquid; γ , γ form, β , β form; M.P., melting point; T, transition.

Thermal Data

Although the tabular method of calculating free energies used for vapor pressures is undoubtedly superior when the necessary data are available, most of the cases to be considered will be dealt with in the analytical form of standard free-energy equation. One may proceed to write such equations in a purely formal way for the different states of manganese, bearing in mind that despite the somewhat unsatisfactory nature of some of the specific-heat data, most cases of interest will involve cancellation (either implicit or explicit) of the uncertain portions.

1. For the transformation,

$$\text{Mn}_C = \text{Mn}_D,$$

$$\Delta C_p = 5.22 - 7.38 \cdot 10^{-3} T,$$

$$\Delta F_{1,108}^0 = 0,$$

$$\Delta H_{1,108} = 148,$$

$$\Delta H = \Delta H_0 + 522 T - 3.69 \cdot 10^{-3} T^2,$$

$$= -1,102 + 5.22 T - 3.69 \cdot 10^{-3} T^2. \quad (\text{Equation 3})$$

$$\Delta F_{1,108}^0 = 0 = -1,102 - 5.22 T \ln T + 3.69 \cdot 10^{-3} T^2 + I T,$$

$$I = +33.52,$$

$$\Delta F^0 T = -1,102 - 5.22 T \ln T + 3.69 \cdot 10^{-3} T^2 + 33.52 T. \quad (\text{Equation 4})$$

2. For the transformation,

$$\text{Mn}_D = \text{Mn}_V,$$

$$\Delta C_p = 1.0,$$

$$\Delta F_{1,317}^0 = 0,$$

$$\Delta H_{1,317} = 244,$$

$$\Delta H = \Delta H_0 + 1.0 T,$$

$$= -1,073 + 1.0 T. \quad (\text{Equation 5})$$

$$\Delta F_{1,317}^0 = -1,073 - 1.0 T \ln T + I T,$$

$$I = +7.99,$$

$$\Delta F_T^0 = -1,073 - 1.0 T \ln T + 7.99 T. \quad (\text{Equation 6})$$

3. For the transformation,

$$\text{Mn}_{\text{V}} = \text{Mn}(\underline{1}),$$

$$\Delta C_p = 1.30,$$

$$\Delta F_{1,493}^0 = 0.$$

$$\Delta H_{1,493} = 3,576,$$

$$\Delta H = \Delta H_0 + 1.30 \underline{T},$$

$$= -5,516 + 1.30 \underline{T}.$$

(Equation 7)

$$\Delta F_{1,493}^0 = 0 = -5,516 - 1.30 \underline{T} \ln \underline{T} + \underline{I} \underline{T},$$

$$\underline{I} = +11.01,$$

$$\Delta F_{\underline{T}}^0 = -5,516 - 1.30 \underline{T} \ln \underline{T} + 11.01 \underline{T}.$$

(Equation 8)

4. For the transformation,

$$\text{Mn}(\underline{1}) = \text{Mn}(\underline{g}),$$

$$\Delta C_p = -6.33,$$

$$\Delta F_{1,784}^0 = -RT \ln \frac{30}{760} = +11,460,$$

$$\Delta F_{2,274}^0 = 0 \text{ (normal boiling point),}$$

$$\Delta F_{1,784}^0 = +11,460 = \Delta H_0 + 6.33 \underline{T} \ln \underline{T} + \underline{I} \underline{T},$$

$$\Delta F_{2,274}^0 = 0 = \Delta H_0 + 6.33 \underline{T} \ln \underline{T} + \underline{I} \underline{T}.$$

Solving simultaneously,

$$\underline{I} = -77.92,$$

$$\Delta H_0 = +65,930,$$

$$\Delta F_{\underline{T}}^0 = +65,930 + 6.33 \underline{T} \ln \underline{T} - 77.92 \underline{T}.$$

(Equation 9)

$$\Delta H = +65,930 - 6.33 \underline{T}.$$

(Equation 10)

5. By adding equations 3 and 5, and 4 and 6, for the transformation,

$$\text{Mn}_{\text{Q}} = \text{Mn}_{\text{V}},$$

$$\Delta H = -2,175 + 6.22 \underline{T} - 3.69 \cdot 10^{-3} \underline{T}^2.$$

(Equation 11)

$$\Delta F_{\underline{T}}^0 = -2,175 - 6.22 \underline{T} \ln \underline{T} + 3.69 \cdot 10^{-3} \underline{T}^2 + 41.46 \underline{T}.$$

(Equation 12)

Similarly, from equations 7 and 11, and 8 and 12, for the transformation,

$$\text{Mn}(\alpha) = \text{Mn}(\underline{1})$$

$$\Delta H = -7,691 + 7.55 \underline{T} - 3.69 \cdot 10^{-3} \underline{T}^2. \quad (\text{Equation 13})$$

$$\Delta F_T^0 = -7,691 - 7.55 \underline{T} \ln \underline{T} + 3.69 \cdot 10^{-3} \underline{T}^2 + 52.47 \underline{T}. \quad (\text{Equation 14})$$

Finally, by the addition of equations 13 and 10, and 14 and 9, for the transformation,

$$\text{Mn}(\alpha) = \text{Mn}(\underline{g}),$$

$$\Delta H = +58,239 + 1.22 \underline{T} - 3.69 \cdot 10^{-3} \underline{T}^2. \quad (\text{Equation 15})$$

$$\Delta F_T^0 = +58,239 - 1.22 \underline{T} \ln \underline{T} + 3.69 \cdot 10^{-3} \underline{T}^2 - 25.45 \underline{T}. \quad (\text{Equation 16})$$

These last equations will be useful in referring reactions involving manganese vapor to the standard state of solid α Mn.

Manganese Ion

The free energy of the manganese ion is a quantity that may be used in calculating the free energy of formation from solubility measurements. Because early experiments on the electrode potential of manganese were unsatisfactory, Lewis and Randall did not include the manganous ion in their resume of standard electrode potentials.

The manganese electrode is very sensitive to oxygen when used in electrolytic cells, and the ordinary form of cast metal shows erratic behavior. Royce and Kahlenberg (321) have, however, obtained consistent behavior, using especial precautions to avoid the presence of oxygen in these cells, and experimenting with various concentrations of manganese amalgam. Their results indicate that amalgams above 1 percent show substantially constant electro motive force in a given concentration of electrolyte. Table 47 shows selected portions of these results and various calculations which have been made from them.

TABLE 47. - Electrode Potential of Manganese at 20°C.

Electro- lyte	Stated normality	<u>m</u> , molality	e.m.f. reported	Cell e.m.f.	e.m.f. v. \underline{H}_2	γ activity coefficient	Correc- tion	E_0 standard electrode potential
MnSO ₄	1.0	0.5	1.4490	0.8890	1.1712	0.110	0.0366	1.1346
do.	.5	.25	1.4530	.8930	1.1752	.157	.0409	1.1343
MnCl ₂	1.0	.5	1.4335	.8735	1.1557	.490	.0178	1.1379
do.	.5	.25	1.4380	.8780	1.1602	.498	.0263	1.1339
Average	---	---	---	---	---	---	---	1.1352 ±0.0011

In this table the figures given by the authors for "normality" seem in fact to be double molalities rather than equivalents per liter solution. Nowhere in their article do the authors state specifically what they mean by "normality", but a clue is furnished by the statement they make that "8-normal" solution is virtually saturated at ordinary temperatures. A saturated solution contains approximately 39 percent MnSO_4 , formula weight 151, density of solution about 1.4. Thus a truly 8-normal solution would contain $\frac{8 \times 75.5}{8 \times 75.5 + 1,000} = 37.8$ percent MnSO_4 , but a 4-molal solution contains $\frac{8 \times 75.5}{8 \times 75.5 + 1,000} = 37.8$ percent MnSO_4 .

The electro motive force reported by the authors is stated to be based on the value +0.560 volt for the normal calomel electrode used as reference standard. To put these results on the basis of the hydrogen electrode, as used by Lewis and Randall (234, p.407), we shall subtract 0.560 and add 0.2822, the latter figure being the electro motive force of the normal electrode selected by these authors. In this way the figures given in the column headed "e.m.f. v. H_2 " are obtained. The activity coefficients in the next column are taken from Landolt-Börnstein (228). Then, since $E = E_0 - \frac{RT}{2F} \ln (\gamma m)$, the correction term represented by the last quantity of this formula may be calculated and the standard electrode potential E_0 found directly.

Inspection of the values of E_0 so obtained shows good consistency between the various measurements. Accepting the average result given, there is obtained for the reaction $\text{Mn(Hg)} \rightarrow \text{Mn}^{++}$,

$$\begin{aligned} 4F_{291}^0 &= -nFE_0 = -2 \times 23,074 \times 1.1352, \\ &= -52,450 \text{ calories.} \end{aligned}$$

Although the indicated deviation from the mean of the above value for E_0 would show an apparent deviation from the mean of 50 calories in free energy, it must not be forgotten that the cells used by Royce and Kahlenberg may have contained liquid-junction potentials of unknown amount that could in extreme cases represent 500 calories or more.

OXIDES OF MANGANESE

Manganous Oxide

This material occurs in nature as the mineral manganosite and so far as is known does not show allotropicism. The density of the natural mineral is approximately 5.1.

Specific Heats at Low Temperatures

The only available information regarding low-temperature specific heats of manganous oxide are the measurements by Millar (257). In his review of low-temperature specific heats, Kelley (200) has corrected Millar's original data slightly and reports $S_{298.1} = 14.4 \pm 0.6$. Millar had calculated $S_{298} = 14.92$.

Kelley's value will be accepted here. Data on compressibility and coefficient of expansion have not been found.

Specific Heats at High Temperatures

Previous to Millar's low-temperature work the only reported specific heat of manganous oxide was that of Regnault (512), who reported a mean value of 11.14 between 13° and 98°C. The true specific heat from Millar's experiments at 0°C. is 10.00, indicating that Regnault's result is too high.

In the absence of data on the specific heat at high temperatures the only estimate possible is a rational guess based on the figure at 0°C. Then it may be assumed that at the melting point of MnO, which has been given by Tiede and Birnbrauer (372) as 1,650°C., a maximum true specific heat of 14 calories per formula weight⁵ has been reached.

Consider now the conventional quadratic expression used by Lewis and Randall and others to represent specific heats at high temperatures

$$\underline{C_p} = \underline{a} + \underline{b}t + \underline{c}t^2,$$

where a, b, and c are numerical coefficients and t the temperature in degrees centigrade. Coefficient a is obviously the value at t = 0, that is, for MnO = 10.00. If the quadratic is differentiated

$$\frac{d\underline{C_p}}{dt} = \underline{b} + 2\underline{c}t,$$

$$\left(\frac{d\underline{C_p}}{dt}\right)_{t=0} = \underline{b},$$

then b is the slope of the specific-heat curve at T = 273.1, and c is determined by the limiting value of C_p = 14 at 1,650°C.

From Millar's specific-heat curve, b = 8.4·10⁻³ is found, and c = -3.62·10⁻⁶ from which

$$\underline{C_p}(\text{MnO}) = 10.00 + 8.4 \cdot 10^{-3} \underline{t} - 3.62 \cdot 10^{-6} \underline{t}^2.$$

Substituting t = T - 273.1,

$$\underline{C_p}(\text{MnO}) = 7.43 + 1.038 \cdot 10^{-2} \underline{T} - 3.62 \cdot 10^{-6} \underline{T}^2.$$

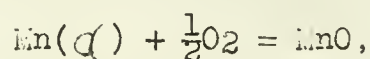
(Equation 17)

⁵ The figure 14 is crudely estimated as follows: C_v for Mn and O may approach the theoretic value of 6 calories each. At high temperatures the specific heats of the compound will be nearly additively the sum of the specific heats of its constituent elements. C_p - C_v is 0.3 at 298.1°K, and because of the hardness of manganese, may be expected to increase fairly rapidly with the temperature. Actual values of C_p tend to drift somewhat higher than 6 calories per gram atom. (C_p - C_v), by analogy with zinc oxide, which has about the same stability as MnO and a formula weight of the same order of magnitude the figure 13.9 is estimated for ZnO near its sublimation point (1,900°C.). Mean specific heat 25° to 1,100°C. (W.P.White), 12.16; true specific heat at 25°C. 9.75. $\frac{19}{11} \times (12.16 - 9.75) + 9.75 = 13.9$.

Sublimation and Vapor Pressures

In spite of the fact that Berthelot (24) and Le Chatelier (230) are in substantial agreement with 90,900 and 90,800 calories for the heat of formation of manganous oxide, the figure selected here is the recent one of Roth (319), who reports $96,500 \pm 650$ calories for the heat of formation at constant pressure.

Then for the reaction



$$S_{298} = 7.3 + 24.5 - 14.4 \quad \Delta S_{298} = 17.4,$$

$$\Delta h_{298} = -96,500,$$

$$\Delta F_{298} = \Delta H - T\Delta S,$$

$$= -91,510,$$

$$\Delta C_p = 0.40 + 2.50 \cdot 10^{-3}T - 3.62 \cdot 10^{-6}T^2,$$

$$-96,500 = \Delta H = \Delta H_0 + 0.4 T + 1.25 \cdot 10^{-3}T^2 - 1.21 \cdot 10^{-6}T^3,$$

$$\Delta H_0 = -96,700,$$

$$-91,310 = \Delta F_{298} = -96,700 - 0.4 T \ln T - 1.25 \cdot 10^{-3}T^2 + 0.60 \cdot 10^{-6}T^3 + I T,$$

$$I = +20.68,$$

$$\Delta F_T^0 = -96,700 - 0.4 T \ln T - 1.25 \cdot 10^{-3}T^2 + 0.60 \cdot 10^{-6}T^3 + 20.68 T. \quad (\text{Equation 18})$$

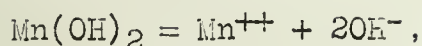
Free Energy of MnO and Mn(OH)₂ from Solubility

The free energy of the Mn^{++} ion obtained in a previous section of this chapter may be used to calculate an approximate value for the free energy of Mn(OH)_2 , using the solubility measurements of Sackur and Fritzmann (332) for this last material. These authors report a solubility of 2.15 moles per liter determined from a conductivity of $1.39 \cdot 10^{-5}$ for the saturated solution in the presence of barium sulphate, and of conductivity water $2 \cdot 10^{-6}$. They measured the conductivity when successive portions of approximately 1/100-normal Ba(OH)_2 solution were added to 1/100-normal MnSO_4 , taking the minimum value so obtained as the lowest observable point. Upon replotting the original data and drawing straight lines through the points nearest the minimum, as seems justified by the data, the figure $1.05 \cdot 10^{-5}$ is obtained for the minimum conductivity. Correcting for the BaSO_4 and conductivity water, the new figure is $6.2 \cdot 10^{-6}$ for the conductivity due to Mn(OH)_2 . Taking 174 for the ion mobility of the OH^- ion and 44 for $\frac{1}{2} \text{Mn}^{++}$, the normal concentration (c) of Mn(OH)_2 is

$$c = \frac{1,000 \cdot 6.2 \cdot 10^{-6}}{218} = 2.84 \cdot 10^{-5},$$

and the molal solubility is $1.42 \cdot 10^{-5}$.

Then for the reaction



the free energy of the reaction is

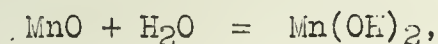
$$\Delta F_{291}^0 = RT \ln (a_{\text{Mn}^{++}}) \cdot (a_{\text{OH}^-})^2,$$

where (Mn^{++}) is $1.42 \cdot 10^{-5}$, and (OH^-) is $2.84 \cdot 10^{-5}$, if it is assumed that the activity coefficient in these very dilute solutions is unity. The numerical result is $\Delta F_{291}^0 = +18,600$. Lewis and Randall give for the free energy of the hydroxide ion -37,455 calories, from which, by summation in accordance with the above reaction, $\Delta F_{291}^0 \text{ Mn(OH)}_2 = -145,960$ calories. It is difficult to estimate the accuracy of this latter figure, but it must be compared to the value -91,310 for MnO, obtained from the third-law calculation.

To make such comparison, an estimate of the entropy of Mn(OH)_2 and of the heat of hydration may be made. For the first item, it may be noted that the entropy of CaO is 9.5, and for Ca(OH)_2 , 17.4, an increase of 8.0 units due to the water. Since Mn(OH)_2 is a much less stable hydroxide than Ca(OH)_2 , it must necessarily be true that the entropy increase will be greater for the former than the latter. On the other hand, the difference cannot be as great as the entropy of ice, which may be taken as 11 units. The estimate may then be set at an increase of 9.5 over the oxide. Since MnO has an entropy of 14.4, the entropy of Mn(OH)_2 is estimated as 23.9 ± 1.5 .

The heat of hydration of MnO must be small and may be zero. Mixter's (261) value of a heat evolution of 4,000 calories cannot be of great accuracy, because the peroxide method has been shown unreliable, but a low value is indicated.

Then for the reaction



$$\begin{array}{cccc} S_{298} & 14.4 & 15.9 & 23.9, \end{array}$$

$$\Delta S = -6.4,$$

$$\Delta H = 0,$$

$$\Delta F_{291} = \Delta H - T \Delta S,$$

$$= +1,860.$$

If the value of free energy of liquid water is taken as -56,560 and for Mn(OH)_2 -145,960, and these values are substituted in appropriate places for the above reaction, the free energy of MnO is calculated at 291°K. as -91,260. In the calculation the difference between entropies at 298° and 291°K. have been ignored in several places, and it is to be regarded as an approximation only.

The concordance is more than could be expected in view of the devious methods used to obtain the two results. It must not be forgotten that the low-temperature specific heats of MnO show a marked anomaly, which makes the entropy calculation uncertain by 200 calories, and Roth's heat of formation is not claimed by him to be closer than 700 calories. These deviations alone might account for differences of several entropy units. Solubility measurements by conductivity are generally high, because of the possibility of unprecipitated impurities in the reagents used. Moreover, the measurements were made at different temperatures, and specific-heat data are lacking to reduce them to a common basis.

Higher Oxides of Manganese

Specific Heats

Manganic Oxide. - The mineral braunite is an impure manganic oxide, crystallizing in the tetragonal system and with a density of approximately 4.8. Thermodynamic data are meager. The only available specific heat is that of Oeberg (280), who gave 0.1620 for the mean specific heat per gram between 15° and 99°C. This may be taken as equivalent to 25.40 per formula weight at 57°C. In this case there are no data for direct determination of the b coefficient, and the best that can be done is to estimate it by analogy with Fe₂O₃. Parks and Kelley (291) have made measurements at low temperatures and find a true specific heat of Fe₂O₃ at 0°C. = 23.45. From the high-temperature specific heats of Roth and Bertram (320) one may estimate C_p (Fe₂O₃ at 47.5°C.) = 25.25. Then b is very nearly $\frac{25.25 - 23.45}{4.75} = 0.0390$.

Furthermore, it will be shown in a succeeding part of this work that b for MnO₂ is $3.09 \cdot 10^{-2}$ and from the MnO curve, b is $0.84 \cdot 10^{-2}$, so that if Mn₂O₃ is considered equal to MnO + MnO₂, b from this source should be $3.93 \cdot 10^{-2}$. Finally, a similar assumption as to the limiting value of C_p , as in the case of MnO, supposes in this case C_p (Mn₂O₃) = 35 at the temperature of dissociation of this material, which may be taken as 1,100°C. Then

$$a + 3.9 \cdot 10^{-2} t_1 + c t_1^2 = 25.40 \quad (t_1 = 57.0),$$

$$a + 3.9 \cdot 10^{-2} t_1 + c t_2^2 = 35.00 \quad (t_2 = 1,100),$$

whence $a = 23.26$, $c = -2.57 \cdot 10^{-5}$,

$$C_p(\text{Mn}_2\text{O}_3) = 23.26 + 3.9 \cdot 10^{-2} t - 2.57 \cdot 10^{-5} t^2,$$

$$\text{or } C_p(\text{Mn}_2\text{O}_3) = 10.33 + 5.3 \cdot 10^{-2} t - 2.57 \cdot 10^{-5} t^2. \quad (\text{Equation 19})$$

Mangano-Manganic Oxide. - The compound Mn₃O₄ exists in nature as the tetragonal crystals of hausmanite and has an approximate density of 4.8. Low-temperature specific heats have been determined by Millar, who reports $S_{298} = 35.73$. Kelley's recalculation of Millar's results is 35.5 ± 0.7 , and this latter figure will be used in future calculations.

The high temperature specific heats may be roughly estimated in the same manner as previously used for MnO . Thus $\underline{a} = \underline{C_p} 273.1 = 32.40$ from a curve of Millar's data, and from the slope of the curve at the same point $\underline{b} = 4.24 \cdot 10^{-2}$. No data as to the melting point are available, but it cannot be far from $1,500^\circ\text{C}$., at which temperature it may be assumed that the specific heat has reached a limiting value of 49 calories. Then

$$32.40 + 4.24 \cdot 10^{-2} \underline{t} + \underline{c} \underline{t}^2 = 49 \quad (\underline{t} = 1,500),$$

$$\underline{c} = -1.82 \cdot 10^{-5},$$

$$\underline{C_p}(\text{Mn}_3\text{O}_4) = 32.40 + 4.24 \cdot 10^{-2} \underline{t} - 2.09 \cdot 10^{-5} \underline{t}^2,$$

$$\text{or } \underline{C_p}(\text{Mn}_3\text{O}_4) = 19.25 + 5.38 \cdot 10^{-2} \underline{T} - 2.09 \cdot 10^{-5} \underline{T}^2. \quad (\text{Equation 20})$$

Manganese Dioxide. - The compound MnO_2 is found in nature as rhombic crystals of pyrolusite, or tetragonal crystals of polianite, density about 4.8. Low-temperature measurements have been made by Russell, (331), who determined a few mean specific heats and more completely by Millar, whose data will be used here. A single mean specific heat from 17° to 48°C . is also reported by Kopp (210), whose result (13.8) virtually coincides with a slight extrapolation of Millar's points.

The entropy from Millar's data is $\underline{S}_{298} = 13.9 \pm 0.4$, the true specific heat at 0°C . 13.00, and $\frac{(\underline{dC_p})}{\underline{dt}} \underline{t}=0 = \underline{b} = 3.09 \cdot 10^{-2}$. Since manganese dioxide dissociates in oxygen at 565°C ., it may be assumed that the limiting value of 21.0 is reached at that point. Then

$$\underline{C_p} = 13.00 + 3.09 \cdot 10^{-2} \underline{t} + \underline{c} \underline{t}^2 = 21.0 \quad (\underline{t} = 565),$$

$$\underline{c} = -2.97 \cdot 10^{-5},$$

$$\underline{C_p}(\text{MnO}_2) = 13.00 + 3.09 \cdot 10^{-2} \underline{t} - 2.97 \cdot 10^{-5} \underline{t}^2,$$

$$\text{or } \underline{C_p}(\text{MnO}_2) = 1.92 + 4.71 \cdot 10^{-2} \underline{T} - 2.97 \cdot 10^{-5} \underline{T}^2. \quad (\text{Equation 21})$$

Thermal and Related Data

General Discussion. - The following figures can be gleaned from the literature on the higher oxides of manganese.

1. MnO_2 : Heat of formation, entropy, standard electrode potential, thermal dissociation at high temperatures.
2. Mn_2O_3 : no direct data except three determinations of thermal dissociation.
3. Mn_3O_4 : Heat of formation and entropy.

Preliminary calculations show that the data are not entirely consistent with each other. Thus, assuming the figures for MnO_2 to be correct and going through the various steps to arrive at the heat and free energy of Mn_3O_4 , the entropy of this last material as calculated from these values differs from the experimental by too large a discrepancy to be ascribable to errors in the entropy determination. Similarly, the converse procedure, starting from Mn_3O_4 , and ending with MnO_2 , leads to figures for free energy and heat of formation of this latter material which differ too greatly for the error in the entropy of MnO_2 .

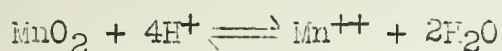
The procedure that seems best adapted to obtain as good thermodynamic concordance as possible is (a) starting from MnO_2 , calculate the properties of Mn_2O_3 , (b) starting from Mn_3O_4 , calculate similar data for Mn_2O_3 , (c) select values for Mn_2O_3 on figures obtained from (a) and (b); (d) using the results of (c), recalculate a revised or adjusted dissociation pressure curve for Mn_3O_4 and MnO_2 .

The new dissociation pressures so calculated will differ from both sets of experimental data on this subject but will distribute the errors somewhat and produce free-energy equations that are consistent.

Manganese Dioxide. - In his paper on low-temperature specific heats, Millar (257) reviewed the heats of formation of MnO_2 (Berthelot 125,300, Le Chatelier 126,000, and Mixter 119,600) and selected Berthelot's figure to calculate the free energy of MnO_2 . Mixter's work was rejected, apparently because Mixter himself was pessimistic about its accuracy. If, however, the heat of formation of MnO_2 is calculated either by means of the standard electrode potential of MnO_2 , or from dissociation pressures as described in the general discussion above, lower values than those of Berthelot or Le Chatelier are obtained, agreeing more nearly with Mixter's figure. It seems better, therefore, to calculate ΔH for MnO_2 , rather than to accept the calorimetric data, especially since this latter type of determination is technically difficult in the case of MnO_2 and would require extremely careful analysis of product when determined in a bomb calorimeter.

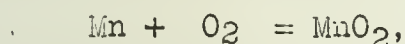
Many attempts to utilize the manganese dioxide electrode for measuring precise electrode potentials have been made, but unfortunately most of these results are not usable because of the irreversibility of the cells, uncertain liquid junctions, and unsuitable selection of electrolytes. However, Brown and Liebafsky (41) have recently published measurements which seem reliable, even though a simultaneously appearing article by Popoff, Reddick, and Becker (307) describes the MnO_2 electrode as irreversible. These latter authors say, with reference to the experiments of the former, "The main argument is the preparation and analysis of the manganese dioxide." In spite of the objections raised, it must be admitted that the history and behavior of the cells set up by Brown and Liebafsky must be regarded as fairly satisfactory, and their result will be accepted here as best available.

According to these latter authors, a value of +1.236 volts was obtained at 25°C. for the standard potential of the reaction



Then ΔF_{298} for the reaction is $-1.236 \times 2 \times 23,074 = -57,050$. Using the value for the free energy of the Mn^{++} ion given in a previous part of this work, $\Delta F_{291}(\text{Mn}^{++}) = -52,450$, and $-56,560$ for liquid water at 25°C., in accordance with Lewis and Randall, the difference between the reference temperature for Mn^{++} and for the reaction may be ignored, and $\Delta F_{298}(\text{MnO}_2)$ calculated from the above reaction as $-108,610$ calories.

Then for the reaction



$$\underline{S} \quad 7.3 \quad 49.0 \quad 13.9 \quad \Delta \underline{S} = -42.4,$$

$$\Delta F_{298} = \Delta H_{298} - T\Delta S,$$

$$-10,861 = \Delta H_{298} + 12,650,$$

$$\Delta H_{298} = -121,260.$$

Thermal dissociation of manganese dioxide. - A number of experimental efforts to determine the dissociation pressure of MnO_2 furnish data that may be used to calculate the heat and free energy of Mn_2O_3 . Thus Meyer and Rötgers (256) report that MnO_2 dissociates into Mn_2O_3 at 530°C. in air and 565°C. in oxygen, but the process was slow and of doubtful reversibility as far as could be determined by their dynamic method. For the dissociation of Mn_2O_3 into Mn_3O_4 , they gave 940 and 1,090° in air and in oxygen, respectively, and were able to show the process reversible. They used MnO_2 prepared from the nitrate, with the correct percentage of active oxygen.

Similarly, Adkenasy and Klonowsky (2) gave a number of points on the dissociation curve disagreeing with Meyer and Rötgers, but they had used precipitated MnO_2 . Further, Kendall and Fuchs (201) report a value for pure MnO_2 agreeing with neither of the previously mentioned experimenters.

Honda and Soné (167) determined critical temperatures of structure change by measuring the variation of magnetic susceptibility at various temperatures. Their figures are close to those of Meyer and Rötgers.

The most extensive work on the dissociation of MnO_2 has been done by Drucker and Hüttner (75). These authors show that the dissociation pressure of manganese oxide depends on the purity and history of the material and is reversible in a limited range of temperatures. They obtained many determinations, of which one set, using precipitated MnO_2 and working with excess oxygen pressure up to the very last portion of a determination, is reported to represent the best measurements. Their experiments show that solid solutions of oxides are

probably not formed, but at low temperatures the water content of hydrated or precipitated MnO_2 cannot be removed without partial decomposition, and at high temperatures the reaction to form Mn_2O_3 becomes irreversible.

In utilizing these various results, recourse may be had to the graphical sigma function method, that is, to make a plot of $\Sigma = -R \ln K + \Delta F_0 \ln T + \frac{1}{2} \Delta F_1 T + \frac{1}{6} \Delta F_2 T^2$, where K is the equilibrium constant or dissociation pressure, R the gas constant, T the absolute temperature, $\Delta F_0, \Delta F_1$, etc., the coefficients of a polynomial representing the change of heat capacity of the reactions. The points of the Σ function plot should be one straight line whose slope is ΔH_0 of the standard free energy equation. The value of integration constant $I = \frac{\Delta H_0}{T} - \Sigma$.

Table 48 shows the results of such calculation for all usable points on the dissociation pressure curve, as given by the investigators mentioned above. Thus, for the reaction



$$\begin{aligned} 2C_p(\text{Mn}_2\text{O}_3) &= 20.66 + 10.3 \cdot 10^{-2} T - 5.14 \cdot 10^{-5} T^2 \\ C_p(\text{O}_2) &= 3.50 + 0.1 \cdot 10^{-2} T, \end{aligned} \quad \begin{array}{l} \text{(From equation 19, p. 114)} \\ \text{(Eastman)} \end{array}$$

$$\begin{aligned} \text{Sum,} \quad & 27.16 + 10.4 \cdot 10^{-2} T - 5.14 \cdot 10^{-5} T^2 \\ 4C_p(\text{MnO}_2) &= 7.68 + 18.84 \cdot 10^{-2} T - 11.88 \cdot 10^{-5} T^2 \quad \text{(From equation 21, p. 115)} \end{aligned}$$

$$\text{Difference, } \Delta C_p = 19.48 - 8.44 \cdot 10^{-2} T + 6.74 \cdot 10^{-5} T^2. \quad \text{(Equation 22)}$$

$$\Sigma = -R \ln p_{\text{atm.}} + 19.48 \ln T - 4.22 \cdot 10^{-2} T + 1.123 \cdot 10^{-5} T^2.$$

The dotted line of fig. 9, which shows graphically the disposition of the various points, turns out to have a slope of +16,160 calories, whence I may be calculated from the points at 631° , 666° , and 696° , which lie on the curve, to be +83.57. Then for the reaction

$$\begin{aligned} \Delta F_T^0 &= 16,160 - 19.48 T \ln T + 4.22 \cdot 10^{-2} T^2 - 1.123 \cdot 10^{-5} T^3 + 83.57 T, \\ \Delta F_{298.1}^0 &= +11,440. \end{aligned} \quad \text{(Equation 23)}$$

$$\begin{aligned} \Delta H_T &= 16,160 + 19.48 T - 4.22 \cdot 10^{-2} T^2 + 2.25 \cdot 10^{-5} T^3, \\ \Delta H_{298.1} &= +18,816. \end{aligned} \quad \text{(Equation 24)}$$

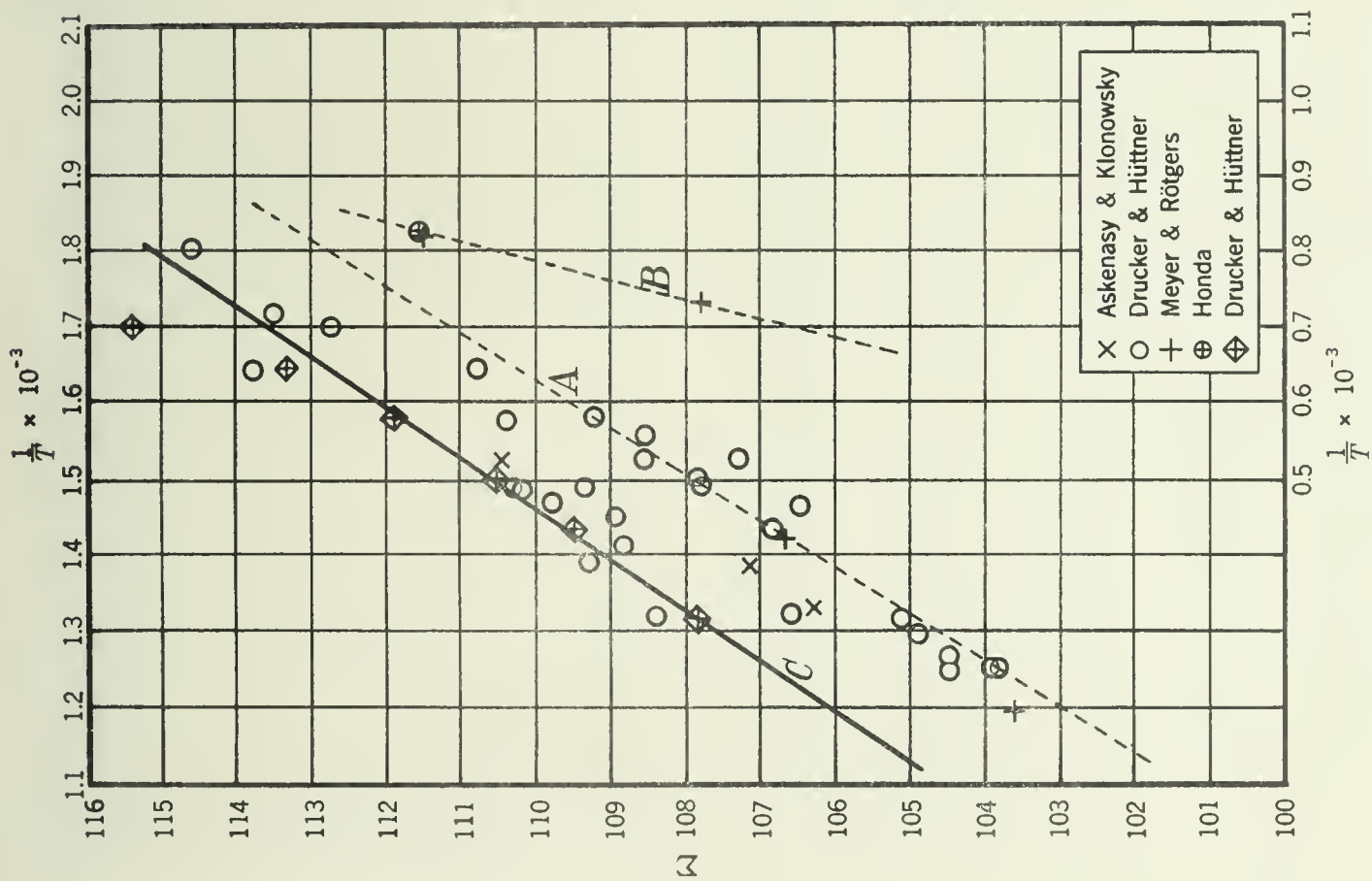


Figure 9.- Sigma function for the dissociation of MnO_2 : A, $MnO_2 - Mn_2O_4$; B, $MnO_2 - Mn_3O_4$; C, $MnO_2 - Mn_2O_3$.

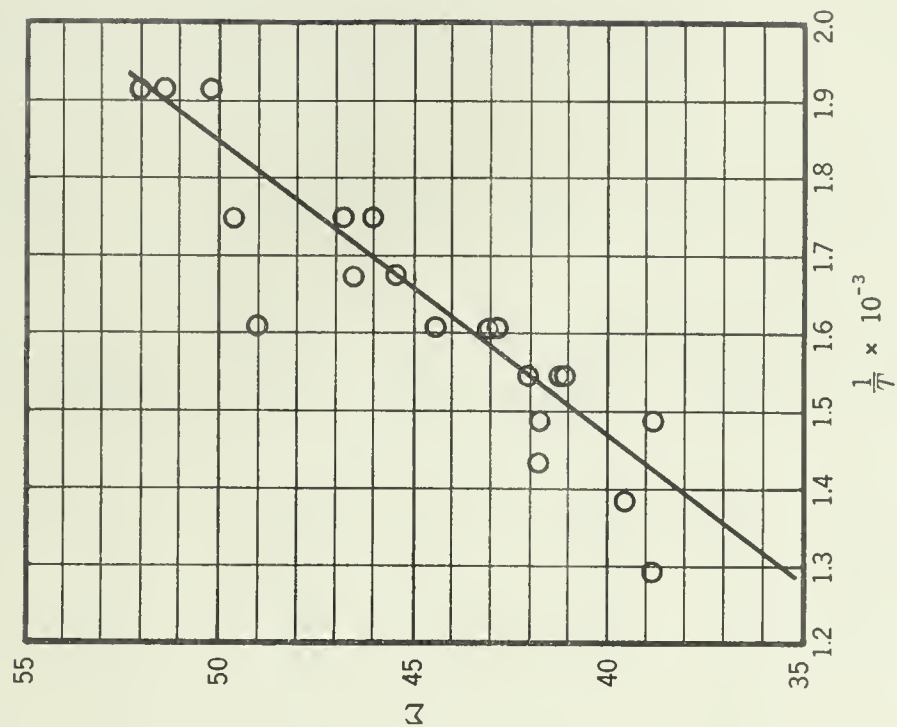


Figure 10.- Sigma function for the reaction $MnCO_3 = MnO + CO_2$.

TABLE 48. - Thermal Dissociation of MnO_2 to Mn_2O_3

Author	Kind of MnO_2	Temperature, °K.	$\frac{1}{T} \cdot 10^3$	P_{mm}	P_{atm}	$-R \ln K$	Σ
Askenasy & Klonowsky	Ppt.....	655	1.527	23.	0.0303	6.955	110.450
do.....	do.....	721	1.387	125.	.165	3.587	107.186
do.....	do.....	751	1.332	200.	.263	2.652	106.278
Drucker & Hüttner	do.....	587	1.704	6.6	.0087	9.435	112.715
do.....	Table 16 ¹ ..	606	1.650	18.4	.0242	7.400	110.759
do.....	do.....	631	1.585	41.3	.0550	5.763	109.201
do.....	do.....	666	1.502	86.2	.1135	4.324	107.839
do.....	do.....	696	1.436	150.3	.1975	3.226	106.804
do.....	do.....	759	1.318	359.7	.4730	1.488	105.105
Meyer & Rötgers	From nitrate....	703	1.423	---	.21	3.102	106.687
do.....	do.....	838	1.194	---	1.00	0.000	103.632
Drucker & Hüttner	Table 5 ¹	675	1.482	25.4	0.0334	6.758	110.289
do.....	" 6....	801	1.249	502.1	.661	.824	104.468
do.....	" 7....	607	1.647	4.0	.0053	10.415	113.772
do.....	do.....	674	1.484	25.0	.0329	6.790	110.310
do.....	do.....	718	1.393	43.6	.0574	5.680	109.276
do.....	do.....	758	1.320	70.1	.0922	4.740	108.374
do.....	do.....	788	1.269	501.5	.659	.828	104.460
do.....	" 8....	581	1.721	4.4	.0058	10.240	113.491
do.....	do.....	633	1.580	23.2	.0305	6.940	110.375
do.....	do.....	671	1.491	41.6	.0548	5.780	109.311
do.....	do.....	682	1.466	47.3	.0623	5.750	109.766
do.....	do.....	689	1.452	51.0	.0671	5.365	108.919
do.....	do.....	707	1.415	54.1	.0712	5.255	108.834
do.....	do.....	756	1.323	252.7	.293	2.440	106.073
do.....	do.....	771	1.297	403.7	.531	1.259	104.894
do.....	do.....	798	1.253	700	.921	.165	103.806
do.....	p 253.....	670	1.493	89.6	.118	4.251	107.780
do.....	do.....	654	1.529	78.3	.103	4.517	107.266
do.....	Table 19 ¹ ..	554	1.806	2.4	.0032	11.460	114.591
do.....	do.....	641	1.560	59.3	.078	5.068	108.541
do.....	do.....	655	1.527	60.2	.079	5.038	108.533
do.....	do.....	683	1.465	174.0	.229	2.933	106.478

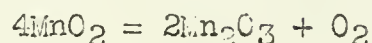
¹ Refers to authors quoted.

$$\text{Since } \Delta F = \Delta H - T\Delta S,$$

$$+11,440 = +18,816 - 298 \cdot \Delta S,$$

$$\Delta S = +24.4.$$

Referring again to the equation of dissociation, and inserting the results of these calculations

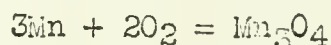


$$\underline{S} = 4 \cdot 13.9 \quad 2 \times \quad 49.0 \quad \Delta S = +24.4 \quad x = 15.5$$

$$\begin{array}{ll} \Delta H & 4(-121,260) = 2\Delta H(\text{Mn}_2\text{O}_3) + 0 \quad \Delta H(\text{react}) = +18,816 \\ & \Delta H_{\text{Mn}_2\text{O}_3} = -233,112 \\ \Delta F & 4(-108,610) = 2\Delta F(\text{Mn}_2\text{O}_3) + 0 \quad \Delta F(\text{react}) = +11,440 \\ & \Delta F_{\text{Mn}_2\text{O}_3} = -211,500 \end{array}$$

As a check on the accuracy of these calculations, consider the entropy of Mn_2O_3 so obtained. Although we have no direct measurements of Mn_2O_3 , its entropy is probably not much different from that of Fe_2O_3 , for which material Kelley gives $S_{298} = 21.5$. The specific heat of Fe_2O_3 at 0°C . is 23.45 and Mn_2O_3 , 23.26; and as the entropies may be taken in rough proportionality, one may estimate $S_{298}(\text{Mn}_2\text{O}_3) = \frac{23.26}{23.45} \times 21.5 = 21.4$, which differs by some 6 units from that obtained from the dissociation, a discrepancy too great even for this approximate method.

Mangano-Manganic Oxide. - Before similar figures are calculated from the dissociation of Mn_2O_3 , the heat and free energy of formation of Mn_2O_4 must be known. Figures for the heat of formation are quoted by Millar (Berthelot and Le Chatelier, both 328,000, Ruff and Gersten, 329,000) but the figure used here will be that of Roth (319), who reports a heat of formation of 345,000 calories, with a stated uncertainty of only 300 calories. Then for the reaction

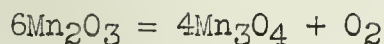


$$\underline{S} = 3 \cdot 7.3 \quad 2 \cdot 49.0 \quad 35.5 \quad \Delta S = -84.4$$

$$\begin{aligned} \Delta F_{298} &= \Delta H - T\Delta S, \\ &= -345,000 + 298.1 \cdot 84.4, \\ &= -319,820. \end{aligned}$$

Dissociation of Manganic Oxide. - The dissociation pressures measured by Meyer and Rötgers have been given above. Honda's point for dissociation in air is 933 compared to 940 for the former authors. To maintain analogy with the MnO_2 dissociation, the Σ function is again calculated, even though the small number of points hardly justify the use of the method. The data are shown in table 49.

Thus, the reaction is



$$4C_p(\text{Mn}_3\text{O}_4) = 77.00 + 21.52 \cdot 10^{-2}T - 8.36 \cdot 10^{-5}T^2 \quad (\text{From equation 20})$$

$$C_p(\text{O}_2) = \frac{6.50 + 0.10 \cdot 10^{-2}T}{\quad} \quad (\text{Eastman})$$

$$\text{Sum} \quad 83.50 + 21.62 \cdot 10^{-2}T - 8.36 \cdot 10^{-5}T^2$$

$$6C_p(\text{Mn}_2\text{O}_3) = 61.98 + 31.80 \cdot 10^{-2}T - 15.42 \cdot 10^{-5}T^2 \quad (\text{From equation 19})$$

$$\text{Difference} = \Delta C_p = 21.52 - 10.18 \cdot 10^{-2}T + 7.06 \cdot 10^{-5}T^2$$

$$\Sigma = -R \ln p_{\text{atm.}} + 21.52 \ln T - 5.09 \cdot 10^{-2}T + 1.177 \cdot 10^{-5}T^2.$$

The slope of the dotted line through the points of the Σ -function plot of figure 10, representing the data on Mn_2O_3 , is $\Delta H_0 = 40,650$, whence $T = 77.97$. Then for the reaction

$$\Delta F_T^0 = 40,650 - 21.52 T \ln T + 5.09 \cdot 10^{-2}T^2 + 1.177 \cdot 10^{-5}T^3 + 77.97 T. \quad (\text{Equation 25})$$

$$\Delta F_{298} = +32,185,$$

$$\Delta H_T = 40,650 + 21.52 T - 5.09 \cdot 10^{-2}T^2 - 2.35 \cdot 10^{-5}T^3. \quad (\text{Equation 26})$$

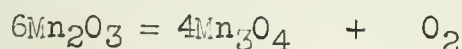
$$\Delta H_{298} = +41,923,$$

$$\Delta S_{298} = \frac{\Delta H - \Delta F}{T} = \frac{9,273}{298} = +31.1.$$

TABLE 49. - Dissociation of Manganic Oxide

Author	Temperature		$\frac{1}{T} \cdot 10^3$	$p_{\text{atm.}}$	$-R \ln K$	Σ
	$^{\circ}\text{C.}$	$^{\circ}\text{K.}$				
Meyer & Rotgers.....	940	1,213	0.8244	0.21	3.102	111.486
do.....	1,090	1,363	.7337	1.00	.000	107.794
Honda.....	933	1,206	.8292	.21	3.102	111.520

Referring again to the decomposition reaction, and inserting the known results above to calculate the properties of Mn_2O_3 , there results

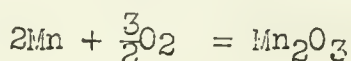


$$S_{298} \quad 6x \quad 4 \cdot 35.5 \quad 49.0 \quad \Delta S = 31.1, \quad x = S_{298}(\text{Mn}_2\text{O}_3) = 26.6$$

$$\begin{aligned} \Delta H \quad 6 \Delta H(\text{Mn}_2\text{O}_3) \quad 4(-345,000) \quad 0 \quad \Delta H(\text{react}) &= +41,923 \\ \Delta H_{298}(\text{Mn}_2\text{O}_3) &= -236,987 \\ \Delta F \quad 6 \Delta F(\text{Mn}_2\text{O}_3) \quad 4(-319,820) \quad 0 \quad \Delta F(\text{react}) &= +32,185 \\ \Delta F_{298}(\text{Mn}_2\text{O}_3) &= -218,577 \end{aligned}$$

The figures so obtained are to be compared with similar ones obtained from the dissociation of MnO_2 . It is apparent that the entropy is now 5.2 units too high, if 21.4 may be assumed correct. The two results for ΔH (-236,987 and -233,112), if averaged, give the figure which will be selected, $\Delta H_{298}(\text{Mn}_2\text{O}_3) = -235,050$, and the average of the ΔF figures (-211,500 and -218,577 is $\Delta F_{298}(\text{Mn}_2\text{O}_3) = -215,040$. Finally, for the formation of Mn_2O_3 , since

$$\Delta S = \frac{\Delta H - \Delta F}{T} = \frac{-20,010}{298} = -67.1,$$



$$S = 2 \cdot 7.3 + \frac{3}{2} \cdot 49.0 \quad x, \quad \Delta S = -67.1$$

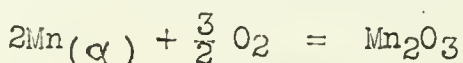
$$x = S_{298}(\text{Mn}_2\text{O}_3) = 21.0.$$

The selection of the average values above results in an entropy of Mn_2O_3 only 0.4 unit different from that estimated from Fe_2O_3 , and represents a degree of thermodynamic consistency that is about all that could be expected in view of the nature of the data.

Standard Free-Energy Equations for Higher Oxides

The information obtained from the rather involved calculations which have just been made enable standard free-energy equations to be constructed for the formation and dissociation of higher oxides which do not lead to thermodynamic inconsistencies, at least as far as entropy is concerned, and which undoubtedly furnish the best interpretation of the data now existing. In view of the experimental difficulties, and the poor reversibility of some dissociation reactions, this interpretation is thought to be better in this respect than the actual determinations made.

Proceeding now to the individual cases, for the reaction



$$C_p(\text{Mn}_2\text{O}_3) = 10.33 + 5.3 \cdot 10^{-2}T - 2.57 \cdot 10^{-5}T^2 \quad (1)$$

$$C_p(2\text{Mn}\alpha) = 7.56 + 1.48 \cdot 10^{-2}T \quad (2)$$

$$C_p(\frac{3}{2}\text{O}_2) = 9.75 + .15 \cdot 10^{-2}T \quad (3)$$

$$\text{Sum of (2 \& 3)} = 17.31 + 1.63 \cdot 10^{-2}T$$

$$\text{Difference} = \Delta C_p = -6.98 + 3.67 \cdot 10^{-2}T - 2.57 \cdot 10^{-5}T^2$$

$$\Delta H_{298} = -235,050 = \Delta H_0 - 6.98 T + 1.83 \cdot 10^{-2}T^2 - 0.857 \cdot 10^{-5}T^2, \quad \Delta H = -234,370,$$

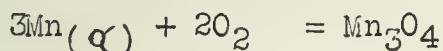
$$\Delta H_T = -234,370 - 6.98 T + 1.83 \cdot 10^{-2}T^2 - 0.857 \cdot 10^{-5}T^2. \quad (\text{Equation 27})$$

$$\Delta F_{298} = -215,040 = -234,370 + 6.98 \, T \ln T - 1.83 \cdot 10^{-2} T^2 + 0.428 \cdot 10^{-5} T^3 + I \, T,$$

$$I = +30.15,$$

$$\Delta F_T^0 = -234,370 + 6.98 \, T \ln T - 1.83 \cdot 10^{-2} T^2 + 0.428 \cdot 10^{-5} T^3 + 30.15 \, T. \quad (\text{Equation 28})$$

Furthermore, from Mn_3O_4 there is obtained



$$\underline{C}_p(\text{Mn}_3\text{O}_4) = 19.25 + 5.38 \cdot 10^{-2} T - 2.09 \cdot 10^{-5} T^2 \quad (1)$$

$$\underline{C}_p(3\text{Mn}\alpha) = 11.34 + 2.21 \cdot 10^{-2} T \quad (2)$$

$$\underline{C}_p(2\text{O}_2) = 13.00 + .20 \cdot 10^{-2} T \quad (3)$$

$$\text{Sum of (2 \& 3)} = 24.34 + 2.41 \cdot 10^{-2} T,$$

$$\text{Difference} = \Delta \underline{C}_p = -5.09 + 2.97 \cdot 10^{-2} T - 2.09 \cdot 10^{-5} T^2,$$

$$\Delta H_{298} = -345,000 = \Delta H_0 - 5.09 \, T + 1.485 \cdot 10^{-2} T^2 - 0.665 \cdot 10^{-5} T^3, \quad \Delta H_0 = -344,627,$$

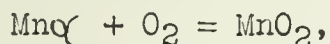
$$\Delta H_T = -344,627 - 5.09 \, T + 1.485 \cdot 10^{-2} T^2 - 0.665 \cdot 10^{-5} T^3. \quad (\text{Equation 29})$$

$$\Delta F_{298} = -319,820 = -344,627 + 5.09 \, T \ln T - 1.485 \cdot 10^{-2} T^2 + 0.322 \cdot 10^{-5} T^3 + I \, T,$$

$$I = +58.35,$$

$$\Delta F_T^0 = -344,627 + 5.09 \, T \ln T - 1.485 \cdot 10^{-2} T^2 + 0.332 \cdot 10^{-5} T^3 + 58.35 \, T. \quad (\text{Equation 30})$$

Finally, for the formation of MnO_2 , according to the reaction



$$\underline{C}_p(\text{MnO}_2) = 1.92 + 4.71 \cdot 10^{-2} T - 2.97 \cdot 10^{-5} T^2 \quad (1)$$

$$\underline{C}_p(\text{Mn}\alpha) = 3.78 + 0.74 \cdot 10^{-2} T \quad (2)$$

$$\underline{C}_p(\text{O}_2) = 6.50 + 0.10 \cdot 10^{-2} T \quad (3)$$

$$\text{Sum of (2 \& 3)} = 10.28 + 0.84 \cdot 10^{-2} T,$$

$$\text{Difference} = \Delta \underline{C}_p = -8.36 + 3.87 \cdot 10^{-2} T - 2.97 \cdot 10^{-5} T^2,$$

$$\Delta H_{298} = -121,260 = \Delta H_0 - 8.36 \, T + 1.94 \cdot 10^{-2} T^2 - 0.99 \cdot 10^{-5} T^3,$$

$$\Delta H_0 = -120,230,$$

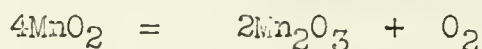
$$\Delta H_T = -120,230 - 8.36 \, T + 1.94 \cdot 10^{-2} T^2 - 0.99 \cdot 10^{-5} T^3. \quad (\text{Equation 31})$$

$$\Delta F_{298} = -108,610 = -120,230 + 8.36 \underline{T} \ln \underline{T} - 1.94 \cdot 10^{-2} \underline{T}^2 + 0.495 \cdot 10^{-5} \underline{T}^3 + \underline{I} \underline{T},$$

$$\underline{I} = -3.31,$$

$$\Delta F_{\underline{T}}^0 = -120,230 + 8.36 \underline{T} \ln \underline{T} - 1.94 \cdot 10^{-2} \underline{T}^2 + 0.495 \cdot 10^{-5} \underline{T}^3 - 3.31 \underline{T}. \quad (\text{Equation 32})$$

In the case of the dissociation of MnO_2 into Mn_2O_3 , the change of specific-heat and preliminary standard free-energy equations have already been given, so there only remains the necessary modification to accommodate the finally selected figures for Mn_2O_3 .



$$\Delta H_{298} \quad 4(-121,260) \quad 2(-235,050) \quad 0 \quad \Delta H = +14,940$$

$$\Delta F_{298} \quad 4(-108,610) \quad 2(-215,040) \quad 0 \quad \Delta F = +4,360$$

$$\Delta H_{298} = +14,940 = \Delta H_0 + 19.48 \underline{T} - 4.22 \cdot 10^{-2} \underline{T}^2 + 2.25 \cdot 10^{-5} \underline{T}^3,$$

$$\Delta H_0 = +12,296,$$

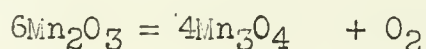
$$\Delta H_{\underline{T}} = +12,296 + 19.48 \underline{T} - 4.22 \cdot 10^{-2} \underline{T}^2 + 2.25 \cdot 10^{-5} \underline{T}^3. \quad (\text{Equation 33})$$

$$\Delta F_{298} = +4,360 = +12,296 - 19.48 \underline{T} \ln \underline{T} + 4.22 \cdot 10^{-2} \underline{T}^2 - 1.123 \cdot 10^{-5} \underline{T}^3 + \underline{I} \underline{T},$$

$$\underline{I} = +72.8,$$

$$\Delta F_{\underline{T}}^0 = +12,296 - 19.48 \underline{T} \ln \underline{T} + 4.22 \cdot 10^{-2} \underline{T}^2 - 1.123 \cdot 10^{-5} \underline{T}^3 + 72.8 \underline{T}. \quad (\text{Equation 34})$$

Similarly, for the reaction



$$\Delta H_{298} \quad 6(-235,050) \quad 4(-345,000) \quad 0 \quad \Delta H = +30,300$$

$$\Delta F_{298} \quad 6(-215,040) \quad 4(-319,820) \quad 0 \quad \Delta F = +10,960$$

$$\Delta H_{298} = +30,300 = \Delta H_0 + 21.52 \underline{T} - 5.09 \cdot 10^{-2} \underline{T}^2 - 2.35 \cdot 10^{-5} \underline{T}^3, \quad \Delta H = +29,027,$$

$$\Delta H_{\underline{T}} = +29,027 + 21.52 \underline{T} - 5.09 \cdot 10^{-2} \underline{T}^2 - 2.35 \cdot 10^{-5} \underline{T}^3. \quad (\text{Equation 35})$$

$$\Delta F_{298} = +10,960 + 29,027 - 21.52 \underline{T} \ln \underline{T} + 5.09 \cdot 10^{-2} \underline{T}^2 + 1.177 \cdot 10^{-5} \underline{T}^3 + \underline{I} \underline{T},$$

$$\underline{I} = +45.67,$$

$$\Delta F_{\underline{T}}^0 = +29,027 - 21.52 \underline{T} \ln \underline{T} + 5.09 \cdot 10^{-2} \underline{T}^2 + 1.177 \cdot 10^{-5} \underline{T}^3 + 45.67 \underline{T}. \quad (\text{Equation 36})$$

If now one proceeds to calculate dissociation pressures for MnO_2 and Mn_2O_3 from equations 54 and 56, the results so obtained at any temperature are much higher than the observed experimental ones. To obtain thermodynamic consistency it has become necessary to assume that the experimental dissociation pressures are incorrect.

The discrepancy so disclosed leads one to a hypothetical assumption that the dissociations actually occurring were not in fact those whose equations were written. Thus, for example, one might suppose that MnO_2 dissociated to MnO , or Mn_3O_4 , instead of Mn_2O_3 .

The necessary calculations for the reaction $2\text{MnO}_2 = 2\text{MnO} + \text{O}_2$ were carried through, assuming the properties of MnO_2 to be correct and calculating from the observed dissociation pressures what the entropy of MnO would be if this actually were the reaction. Without reproducing here the actual figures involved, it may merely be stated that the entropy of MnO_2 calculated in this way turns out to be less than 2 units, an obviously impossible condition.

In contrast to this, calculations for the second hypothesis show an almost satisfactory condition. In the reaction



$$C_p(\text{Mn}_3\text{O}_4) = 19.25 + 5.38 \cdot 10^{-2} T - 2.09 \cdot 10^{-5} T^2$$

$$C_p(\text{O}_2) = 6.50 + 0.10 \cdot 10^{-2} T$$

$$\text{Sum} = 25.75 + 5.48 \cdot 10^{-2} T - 2.09 \cdot 10^{-5} T^2$$

$$C_p(3\text{MnO}_2) = 5.76 + 14.13 \cdot 10^{-2} T - 8.91 \cdot 10^{-5} T^2$$

$$\text{Difference} = \Delta C_p = 19.99 - 8.65 \cdot 10^{-2} T + 6.82 \cdot 10^{-5} T^2$$

The Σ function is then

$$\Sigma = -R \ln p + 19.99 \ln T - 4.325 \cdot 10^{-2} T + 1.137 \cdot 10^{-5} T^2.$$

Table 50 shows the individual points for the "best" results of Drucker and Huttner (75). The slope of the middle points which more closely represent a straight line is $\Delta H_0 = +14,930$. Then

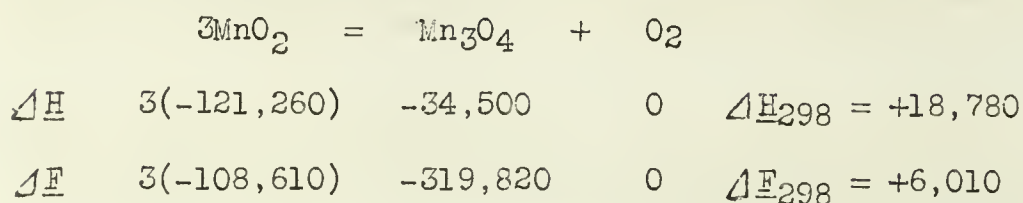
$$\Delta H_T = 14,930 + 19.99 T - 4.325 \cdot 10^{-2} T^2 + 2.27 \cdot 10^{-5} T^3,$$

$$\Delta H_{298} = +18,649,$$

$$\Delta F_T^0 = +14,930 - 19.99 T \ln T + 4.325 \cdot 10^{-2} T^2 - 1.137 \cdot 10^{-5} T^3 + 88.18 T,$$

$$\Delta F_{298} = +10,842.$$

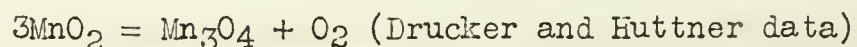
If similar figures are calculated from the accepted heats and free energies of the components



These values, +18,780 and +6,010 calories for heat and free energy, compare not too badly with +18,649 and +10,842, calculated from the assumed dissociation of MnO_2 in Mn_3O_4 , using dissociation pressures, and because the heats of reaction check so well there is a considerable incentive to assume that MnO_2 really dissociated into Mn_3O_4 in the decomposition experiments. However, Drucker and Huttner made X-ray measurements of pure Mn_2O_3 and of their partly dissociated dioxide, and their evidence seems to prove very definitely that not Mn_3O_4 , but Mn_2O_3 , is formed. It is unfortunate that they did not also determine the X-ray pattern of Mn_3O_4 , since such tests would have definitely settled the matter. It should also be stated, however, that if the values 18,780 and 6,010, calculated from MnO_2 and Mn_3O_4 , are used to set up a free-energy equation, the hypothetical decomposition pressures so calculated are considerably higher than the observed. The multiplicity of atoms and molecules involved renders such calculation very sensitive to small errors, but the behavior of the MnO_2 in the dissociation experiments indicates that the true dissociation pressure may be higher than that measured, because it was found that even a small amount of decomposition lowered the observed pressure.

In conclusion, it must be admitted that the question of the dissociation of the higher oxides of manganese is still in a very unsatisfactory condition from a thermodynamic viewpoint. Some crucial experiment which will definitely confirm one of the various hypotheses that have been advanced is greatly needed. Until that time, perhaps the best that can be done is to assume that the dissociation experiments are unreliable and that the processes are probably irreversible.

TABLE 50. - Σ Function for Reaction



Temperature, °K.	$\frac{1}{T} \cdot 10^3$	$-R \ln p$	Σ
587	1.704	9.435	115.394
606	1.650	7.400	113.340
631	1.585	5.763	111.882
666	1.502	4.324	110.514
696	1.436	3.226	109.489
759	1.318	1.488	107.838

MANGANESE SULPHIDE

Physical Properties

Manganese sulphide occurs in nature as the mineral alabandite and crystallizes in the cubic system; density, about 4.0. According to Shibata (346) it melts at $1,610 \pm 3^\circ \text{C}$.

Specific Heats

The only available specific heats for manganese sulphide at low temperatures are those of Anderson (6), who gives for the entropy $S_{298}(\text{MnS}) = 18.7 \pm 0.3$. Manganese sulphide has a very pronounced and peculiarly shaped anomaly in the specific heat curve in the neighborhood of $125^\circ\text{--}140^\circ\text{K}$. Nothing is known as yet concerning the validity of entropy data obtained by determining the anomalous area. Since MnO , MnO_2 , and MnS all have "humps" in the specific heat at low temperatures, these materials offer an excellent opportunity to check the effect of such peculiarities if adequate chemical data could be secured.

The only figure for high-temperature specific heats of MnS is that of Stella (355), who gives 0.1392 per gram, or 12.1 per gram-formula weight of MnS . Under these circumstances the best that can be done for high-temperature specific heats of MnS is an estimate similar in method to that used for the oxides of manganese.

Thus by plotting Anderson's data in the neighborhood of 0°C .,

$$\underline{a} = \underline{C}_p \text{ at } 0^\circ\text{C} = 11.82, \quad \underline{b} = \left(\frac{d\underline{C}_p}{dt} \right)_{t=0} = 5.24 \cdot 10^{-3}, \quad \underline{C}_p \text{ at } 1,610 = 14.0, \\ \underline{c} = -2.42 \cdot 10^{-6}, \text{ and}$$

$$\underline{C}_p(\text{MnS}) = 11.82 + 5.24 \cdot 10^{-3} \underline{t} - 2.42 \cdot 10^{-6} \underline{t}^2,$$

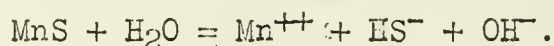
$$\text{or } \underline{C}_p(\text{MnS}) = 10.21 + 6.56 \cdot 10^{-3} \underline{T} - 2.42 \cdot 10^{-6} \underline{T}^2. \quad (\text{Equation 37})$$

Thermal and Related Data

Older determinations of the heat of formation of MnS obtained by precipitation methods are those of Thomsen (369), who reported 44,390 calories for a hydrous sulphide from sulphate solution, and Berthelot (24), who obtained 45,600 calories for the sulphide precipitated from the acetate solution. These values are in complete disagreement with the direct determinations of Wologdine and Penkiewitsch (395), who attempted to measure directly the heat of formation by igniting a mixture of powdered manganese and sulphur in a bomb under nitrogen pressure and obtained 62,900 calories. Many recent investigators have shown that nitrogen combines directly with manganese even at relatively low ignition temperatures, but even this obvious source of error in these experiments cannot account for the large discrepancy. Reference to their original article discloses a none too careful method of analysis of resulting sulphide and considerable vagueness in regard to the rather large corrections applied for the igniting powder of aluminum and potassium chlorate.

By thermodynamic means, values for the heat of formation of MnS may be calculated from other data, which confirm the magnitude of the results of Thomsen and Berthelot; in furtherance of these calculations the value of Berthelot is accepted tentatively.

As first attempt to calculate the heat of formation of MnS, consider the solubility data of Weigel (382), who obtained $54.5 \cdot 10^{-6}$ moles per liter for the solubility from conductivity experiments at 18°C . To calculate the solubility from his measurements of conductivity, Weigel assumed complete hydrolysis of the MnS, but it seems more probable that the actual solution may be better expressed by the equation



He made some measurements of the conductivity of H_2S solutions to show that his calculation was correct, but a simple calculation shows that the HS^- ion exists in relatively large proportion. Thus, if the molal concentration of $\text{Mn}(\text{OH})_2$ from the hydrolysis of sulphide were $5.45 \cdot 10^{-5}$, as determined by Weigel, then (OH^-) is approximately $1 \cdot 10^{-4}$, and since $(\text{H}^+)(\text{OH}^-) = 10^{-14}$, $(\text{H}^+) = 10^{-10}$, the equilibrium constant for the first dissociation of $\text{H}_2\text{S} \rightarrow \text{H}^+ + \text{HS}^-$ is roughly 10^{-7} , then

$$\frac{(\text{H}^+)(\text{HS}^-)}{(\text{H}_2\text{S})} = 10^{-7}, \quad \frac{(\text{HS}^-)}{\text{H}_2\text{S}} = \frac{10^{-7}}{10^{-10}} = 10^3.$$

In view of these conditions, the measurements of Weigel will be recalculated.

It was found that the increase of conductivity due to MnS (green) was $23.38 \cdot 10^{-6}$; then the molal concentration of MnS, assuming complete dissociation into Mn^{++} , HS^- , and OH^- ions, is

$$c = \frac{1,000 \cdot 23.38 \cdot 10^{-6}}{2.45 + 174 + 62} = 7.16 \cdot 10^{-5},$$

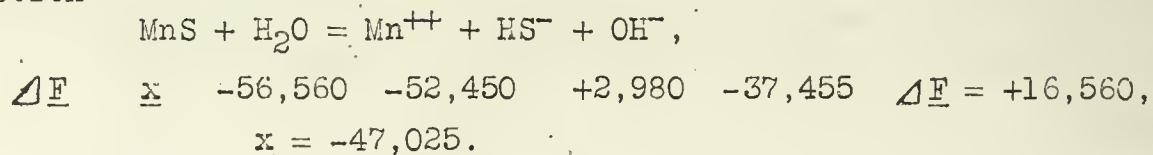
where 45 has been taken for the ion mobility of $\frac{1}{2} \text{Mn}^{++}$, 174 for OH^- , and 62 for HS^- . (Note that the value $7.16 \cdot 10^{-5}$ refers to the green sulphide, and it is only fortuitous that Weigel calculated $71.60 \cdot 10^{-6}$ for the flesh-colored sulphide.

Then for the reaction attending solution of the MnS,

$$\begin{aligned} K &= (\text{Mn}^{++})(\text{HS}^-)(\text{OH}^-) = (7.16 \cdot 10^{-5})^3, \\ &= 3.67 \cdot 10^{-13}, \end{aligned}$$

$$\begin{aligned} \text{and } \Delta F &= -RT \ln K = -4.578 \cdot 291 \cdot (-12.435), \\ &= +16,560. \end{aligned}$$

Further, if one ignores the difference in reference temperatures, and again writes the reaction



The values for H_2O , HS^- , and OH^- are those of Lewis and Randall, that for Mn^{++} being given in an earlier part of this work.

Finally, assuming that -47,025 is the free energy at 298, one may write

$$\text{Mn}(c) + \text{S}(\text{rh}) = \text{MnS}$$

$$S_{298} \quad 7.3 \quad \quad 7.6 \quad \quad 18.7 \quad \Delta S = +3.8$$

$$\Delta F = \Delta H - T\Delta S,$$

$$-47,025 = \Delta H - 1,135,$$

$$\Delta H = -45,890.$$

This result must be regarded as a fortuitous check with Berthelot's -45,600 but confirms the order of magnitude.

As a second check, the work of Guntz (129) may be used. This experimenter found that pyrophoric manganese reacted with SO_2 to give MnS and MnO and was converted to manganese hydroxide with the evolution of 98,600 calories. If from this last figure one subtracts Roth's heat of formation of oxide, 96,500, a "correction" term of 2,100 is found for the "pyrophoric" manganese, if the heat of hydration of the oxide is ignored. So in the reaction

$$3\text{Mn}(\text{pyrc}) + \text{SO}_2 = \text{MnS} + 2\text{MnO},$$

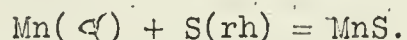
$$\Delta H \quad 3 \cdot 2,100 \quad -70,940 \quad \times \quad 2(-96,500) \quad \Delta H = -82,400$$

$$\Delta H(\text{MnS}) = x = -46,160.$$

In the equation above, the SO_2 value is a recent one of Eckman and Rossini (80). Here again a very satisfactory confirmation of Berthelot's older value for MnS is found.

As a final check on the heat of formation of MnS , reference may be made to some experiments of Jellinek and Von Podjaski (179). These experiments will be considered in detail later in connection with the thermodynamic properties of manganese chloride. Mention will be merely made here that the slopes of their equilibrium data lead to a value of heat of reaction which could only be obtained from a value of heat of formation of MnS corresponding to the figures given above by Berthelot.

In writing the standard free-energy equation for the formation of MnS , the value $\Delta H_{298} = -45,890$ will be taken, as representing the mean of the three good results discussed above. Then $\Delta F_{298} = -47,025$. Therefore the reaction



$$C_p(\text{MnS}) = 10.21 + 6.56 \cdot 10^{-3} T - 2.42 \cdot 10^{-6} T^2$$

$$C_p(\text{Mn } \alpha) = 3.78 + 7.38 \cdot 10^{-3} T$$

$$\frac{C_p(\text{S rh})}{\text{Sum}} = \frac{4.12 + 4.70 \cdot 10^{-3} T}{7.90 + 12.08 \cdot 10^{-3} T} \quad (\text{Lewis \& Randall})$$

$$\text{Difference} = \Delta C_p = 2.31 - 5.52 \cdot 10^{-3} T - 2.42 \cdot 10^{-6} T^2$$

$$\Delta H_{298} = -45,890 = \Delta H_0 + 2.31 T - 2.76 \cdot 10^{-3} T^2 - 0.81 \cdot 10^{-6} T^3. \quad (\text{Equation 38})$$

$$\Delta H_0 = -46,313,$$

$$\Delta F_{298}^0 = -47,025 = -46,313 - 2.31 T \ln T + 2.76 \cdot 10^{-3} T^2 + 0.403 \cdot 10^{-6} T^3 + I T,$$

$$I = +9.90,$$

$$\Delta F_T^0 = -46,313 - 2.31 T \ln T + 2.76 \cdot 10^{-3} T^2 + 0.403 \cdot 10^{-6} T^3 + 9.90 T. \quad (\text{Equation 39})$$

Liquid Manganese Sulphide

There are no direct determinations of the heat of fusion of MnS, but since the important metallurgical use of manganese as a desulphurizer involves liquid sulphide, it may be worth while to make a very approximate estimate of the properties of liquid MnS.

If one attempts to calculate the heat of fusion from the melting points of the FeS - MnS system, using the data of Shibata (346), absurd results are obtained.

As an alternative, the assumption may be made that the entropy of fusion of MnS is the same as FeS. For the latter material, there is available a single approximate measurement of Bornemann and Hengstenberg (30), corresponding to 60 calories per gram or 5,270 calories per FeS. Since the melting point is 1,173°C. the entropy of fusion is 3.64 units; and the heat of fusion of MnS, melting at 1,610°C. is estimated at 6,850 calories per MnS.

A very crude guess at the specific heat of liquid MnS would be 16 calories per formula weight.

MANGANESE CARBONATE

Manganese carbonate occurs in nature as the mineral rhodochrosite, with a density of approximately 3.5, crystallizing in rhombohedra isomorphous with calcite. No allotropic forms have been reported.

Specific Heats

The specific heats of the natural mineral have recently been determined throughout the low-temperature range by Anderson (7), who reports for the entropy $S_{298} = 20.52 \pm 0.4$.

For the high-temperature range no data are available, and the specific heat must be estimated in the same manner as used previously in this work. Thus, from the value taken from Anderson's curve at

$$t = 0^\circ\text{C.}, \underline{a} = \underline{C_p} 273.1 = 18.62, \underline{b} = \left(\frac{d\underline{C_p}}{dT} \right)_{T=273.1} = 3.72 \cdot 10^{-2}.$$

Further, since MnCO_3 reaches a dissociation pressure of 1 atmosphere near 500°C. and assuming that $\underline{C_p} = 35$ at that temperature,

$$\underline{c} = -0.9 \cdot 10^{-5}.$$

$$\text{Then } \underline{C_p} \text{MnCO}_3 = 18.62 + 3.72 \cdot 10^{-2} \underline{t} - 0.9 \cdot 10^{-5} \underline{T}^2,$$

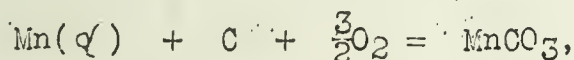
$$\text{or } \underline{C_p}(\text{MnCO}_3) = 7.79 + 4.21 \cdot 10^{-2} \underline{T} - 0.9 \cdot 10^{-5} \underline{T}^2.$$

Thermal and Dissociation Data

The older value of Thomsen for the heat of formation of MnCO_3 , 210,840 calories, is regarded as much less certain than the recent results of Roth (319), who gives for the formation from its elements, $219,100 \pm 800$ calories, and for the reaction



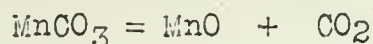
The free energy of MnCO_3 may be calculated from the third law as follows:



$$\underline{S}_{298} \quad 7.3 \quad 1.3 \quad 73.5 \quad 20.5 \quad \Delta \underline{S} = 61.6$$

$$\begin{aligned} \Delta F_{298}^0 &= \Delta H_{298} - T \Delta S_{298}, \\ &= -219,100 + 18,220, \\ &= -200,880. \end{aligned}$$

The figure so obtained may be checked by comparison with dissociation data for MnCO_3 , which have been determined experimentally by Manchot and Lorenz. As before, it will be desirable to set up a Σ function plot for the experimental data. Thus for the reaction



$$\underline{C_p}(\text{MnO}) = 7.43 + 1.04 \cdot 10^{-2} \underline{T} - 0.362 \cdot 10^{-5} \underline{T}^2$$

$$\underline{C_p}(\text{CO}_2) = 7.70 + 0.53 \cdot 10^{-2} \underline{T} - 0.083 \cdot 10^{-5} \underline{T}^2$$

$$\text{Sum} \quad 15.13 + 1.57 \cdot 10^{-2} \underline{T} - 0.445 \cdot 10^{-5} \underline{T}^2$$

$$\underline{C_p}(\text{MnCO}_3) = 7.79 + 4.21 \cdot 10^{-2} \underline{T} - 0.90 \cdot 10^{-5} \underline{T}^2$$

$$\text{Difference} = \Delta \underline{C_p} = 7.34 - 2.64 \cdot 10^{-2} \underline{T} + 0.455 \cdot 10^{-5} \underline{T}^2$$

$$\Sigma = -R \ln p + 7.34 \ln T - 1.32 \cdot 10^{-2} \underline{T} + 0.076 \cdot 10^{-5} \underline{T}^2.$$

Table 51 shows the specific figures based on the various experiments of Manchot and Lorenz, and the corresponding chart of $\sum v. \frac{1}{T}$ appears in figure 10. These authors showed that the equilibrium between carbonate, oxide, and CO_2 was only moderately reversible.

TABLE 51. - \sum Function for Thermal Dissociation of MnCO_3

$$(\sum = -R \ln p + 7.34 \ln T - 1.32 \cdot 10^{-2} T + 0.76 \cdot 10^{-6} T^2)$$

Series	Temperature,		$\frac{1}{T} \cdot 10^3$	P_{mm}	P_{atm}	$-R \ln p$	\sum
	$^{\circ}\text{C.}$	$^{\circ}\text{K.}$					
A.....	250	523	1.913	1.6	0.0021	+12.25	51.45
	300	573	1.746	4.0	.0053	+10.42	49.69
	350	623	1.606	5.5	.0072	+9.79	49.08
	500	773	1.294	855.	1.125	-.23	38.81
B.....	250	523	1.913	3.	.0040	+11.02	50.22
	300	573	1.746	25.	.0329	+6.79	46.06
	325	598	1.672	35.	.0461	+6.12	45.45
	350	623	1.605	127.	.167	+3.56	42.85
	375	648	1.544	286.	.376	+1.95	41.25
	400	673	1.486	855.	1.125	-.23	38.81
C.....	350	623	1.605	115.	.151	+3.75	43.04
	375	648	1.544	303.	.399	+1.83	41.13
D.....	250	523	1.913	7.5	.0099	+9.18	52.06
	300	573	1.746	17.0	.0224	+7.56	46.83
	325	598	1.672	20.0	.0263	+7.23	46.56
	350	623	1.605	57.0	.0750	+5.15	44.44
	375	648	1.544	113.	.149	+3.79	43.09
	400	673	1.486	198.	.261	+2.67	41.71
	425	698	1.432	221.	.291	+2.45	41.71
	450	723	1.384	650.	.856	+0.31	39.53

The line drawn in the figure has this slope, passing through those points which seem from the experimental data to be best. Then the value of I for the free-energy equation is found to be -0.1, and the free energy of dissociation is given by the expression

$$\Delta F_T^0 = +27,245 - 7.34 T \ln T + 1.32 \cdot 10^{-2} T^2 - 0.76 \cdot 10^{-6} T^3 - 0.1 T,$$

$$\Delta F_{298}^0 = +15,913.$$

This final result may be compared with that calculated directly



$$\Delta F_{298} \quad -200,880 \quad -91,310 \quad -93,650 \quad \Delta F_{298} = +15,920.$$

The concordance here between the two values of free energy is, of course, much better than could have been expected from the nature of the equilibrium data.

As a final check on the free energy of formation of MnCO_3 , one may utilize the measurements of the solubility of this material in solutions containing carbon dioxide. Thus, Haehnel (142) found that when the partial pressure of CO_2 was 1 atmosphere, 0.04 percent MnCO_3 was dissolved at 18°C ., and when the pressure was 56 atmospheres, 0.08 percent was so dissolved. The corresponding molal concentrations of MnCO_3 are $3.48 \cdot 10^{-3}$, and $6.96 \cdot 10^{-3}$. Further, one may write $K = \frac{\alpha(\text{MnHCO}_3)^2}{(\text{H}_2\text{CO}_3)^3}$, where α is the dissociation of manganous bicarbonate. In

this case, because of the low concentrations involved, α may be assumed equal to unity. The solubility product of MnCO_3 , $K_3 = (\text{Mn}^{++})(\text{CO}_3^{--})$ is related to K by the expression $K_3 = \frac{K^3(4K_2)}{K_1}$, where K_1 and K_2 are the first and second dissociation constants for carbonic acid. Lewis and Randall give $K_1 = 350 \cdot 10^{-7}$, $K_2 = 3.7 \cdot 10^{-11}$ at 25° . We shall have to assume that the difference between the ratio $\frac{4K_2}{K}$ at 25 and 18° is negligible, since more specific information is lacking. The solubility of CO_2 at 18° is 0.0414 molal when the pressure is 1 atmosphere and 1.25 molal at 56 atmospheres.

$$\text{Then } K(1 \text{ atm}) = \frac{(3.48 \cdot 10^{-3})^2}{(4.14 \cdot 10^{-2})^{1/3}} = 3.5 \cdot 10^{-5},$$

$$\text{and } K(56 \text{ atm}) = \frac{(6.96 \cdot 10^{-3})^2}{(1.25)^{1/3}} = 4.48 \cdot 10^{-5}.$$

$$\text{Further, } K_3(1 \text{ atm}) = \frac{(3.5 \cdot 10^{-5})^3 \cdot 14.8 \cdot 10^{-11}}{350 \cdot 10^{-7}},$$

$$= 1.81 \cdot 10^{-19},$$

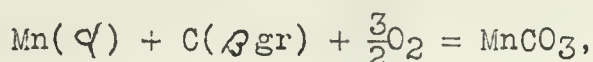
$$K_3(56 \text{ atm}) = \frac{(4.48 \cdot 10^{-5})^3 \cdot 14.8 \cdot 10^{-11}}{350 \cdot 10^{-7}},$$

$$= 4.66 \cdot 10^{-19}.$$

For the reaction $\text{MnCO}_3 = \text{Mn}^{++} + \text{CO}_3^{--}$,

$$\Delta F_{291} = -R T \ln K_3 = +24,950 \text{ at 1 atmosphere and } +24,400 \text{ at 56 atmospheres}$$

If the difference in free energy of Mn^{++} ion and CO_3^{--} ion between 18° and 25°C . is neglected, the value -52,450 may be used for the former, and from Lewis and Randall -125,760 for the latter, then ΔF_{291}^0 for MnCO_3 , from the above reaction becomes, for the two sets of data, -203,160, and -202,610. Correction to 298° , in accordance with the approximate specific heats for the reaction



$$\Delta C_p (\text{near } 25^\circ\text{C}.) = +1.35,$$

reduces these values approximately 500 calories, so that we have $\Delta F_{298}^0 (\text{MnCO}_3) = -202,660$, and -202,110. These figures agree surprisingly well with the previous one, -200,880, obtained from the third law, especially when the many

approximations in the calculation from solubility data are considered, as well as the possible uncertainty of at least 800 calories in the heat of formation of the carbonate.

MANGANESE SULPHATE

Although manganese sulphate occurs in several hydrated forms in nature, these materials have little significance for metallurgical purposes. The present discussion is limited to the anhydrous sulphate. The density of the anhydrous salt is probably near 3.25.

Specific Heats

No specific heats of manganous sulphate at low temperatures are available, and the entropy cannot be determined directly. The only available figure for specific heat is the very old one of Pape (287), who reported 0.182 per gram, or 27.5 per MnSO_4 , as mean heat from 20° to 100°C . Friedrich and Blicke (99) reported that the sulphate melts at 700°C ., but Hofman and Wanjukow (165) found no evidence for fusion at temperatures above 750°C . The pressure of thermal decomposition becomes an atmosphere at $1,020^\circ - 1,030^\circ\text{C}$.

The value of 7 calories per gram-atom, which was selected as the limiting specific heat for the oxides, may be assumed to be reached at $1,027^\circ\text{C}$.

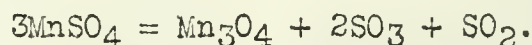
$$\begin{aligned}\text{Then } C_p(\text{MnSO}_4) &= 26.6 + 1.5 \cdot 10^{-2} t, \\ C_p(\text{MnSO}_4) &= 22.5 + 1.5 \cdot 10^{-2} T.\end{aligned}$$

The question of possible allotropic modifications of anhydrous MnSO_4 is one that cannot be settled definitely by the data now available. Thus Friedrich and Blicke, studying heating curves, found arrest points at 700° and 860°C . Hofman found similar points but ascribed the effect to decomposition. In discussing the thermal decomposition of MnSO_4 , it will be shown later that an anomalous condition prevails at 880°C .

Honda and Soné (167) made measurements of magnetic susceptibility between -200° and $1,200^\circ\text{C}$. Their data show a slight change of trend of the curve showing permeability v. temperature at slightly above 700°C . and a very marked change near 850°C .

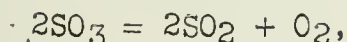
Thermal Decomposition

By far the best data on the thermal decomposition of MnSO_4 are those of Marchal (241). The sulphate decomposes into a mixture of SO_2 , SO_3 , and O_2 , with Mn_3O_4 as the solid phase. It may be supposed that the primary reaction is



The sulphur trioxide so formed dissociates further into SO_2 and O_2 , and the amounts of the various gaseous constituents may be determined if it is assumed

that equilibrium is attained in the reaction .

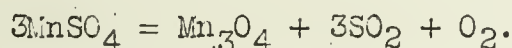


as well as in the primary reaction.

To calculate $\Delta \underline{C}_p$ for the dissociation reaction, a special expedient must be used, because no data are available for the specific heat of SO_3 . However, Lewis and Randall have shown that if $\Delta \underline{C}_p$ is assumed zero, for the reaction



a remarkably consistent value is obtained for the integration constant of the free-energy equation for this latter reaction. Then $\Delta \underline{C}_p$ must be the same for the primary reaction as for the sum of the primary reaction and the SO_3 dissociation, that is,



Then the specific heats for this reaction are given by the following expressions:

$$\underline{C}_p(\text{Mn}_3\text{O}_4) = 19.25 + 5.38 \cdot 10^{-2} \underline{T} - 2.09 \cdot 10^{-5} \underline{T}^2$$

$$\underline{C}_p(3\text{SO}_2) = 23.10 + 1.59 \cdot 10^{-2} \underline{T} - 0.25 \cdot 10^{-5} \underline{T}^2$$

$$\underline{C}_p(\text{O}_2) = \frac{6.50 + 0.10 \cdot 10^{-2} \underline{T}}{1}$$

$$\text{Sum} \quad 48.85 + 7.07 \cdot 10^{-2} \underline{T} - 2.34 \cdot 10^{-5} \underline{T}^2$$

$$\underline{C}_p(3\text{MnSO}_4) = \frac{667.5 + 4.50 \cdot 10^{-2} \underline{T}}{1}$$

$$\Delta \underline{C}_p = -18.65 + 2.57 \cdot 10^{-2} \underline{T} - 2.34 \cdot 10^{-5} \underline{T}^2$$

$$\Sigma = -R \ln K - 18.65 \ln \underline{T} + 1.285 \cdot 10^{-2} \underline{T} - 0.39 \cdot 10^{-5} \underline{T}^2.$$

Table 52 gives the data of Marchal and the corresponding figures from which Σ is determined. Figure 11 shows the results graphically as $\Sigma v. \frac{1}{\underline{T}}$.

Reference to the figure shows that except at the ends of the chart good concordance with the required straight line is obtained. At low temperatures the points deviate markedly, however. If a line (shown dotted) is passed through the two lowest points, it intersects the full line at a temperature of 878° , which is approximately the arrest points 850 and 860, obtained by other investigators mentioned above. The slope of the full line, that is, $\Delta \underline{H}_0$ is +153,800 calories and of the dotted one 221,000. These figures may be used to calculate the heat of formation of MnSO_4 in order to determine the validity of the data.

TABLE 52. - \sum Function for Reaction $\text{MnSO}_4 = \text{Mn}_2\text{O}_4 + 2\text{SO}_3 + \text{SO}_2$

or Its Equivalent, $\text{MnSO}_4 = \text{Mn}_2\text{O}_4 + 3\text{SO}_2 + \text{O}_2$

Temperature °C.	Temperature °K.	$\frac{1}{T} \cdot 10^3$	Δ_{atm}	Δ_{SO_3}	Δ_{SO_2}	K	$-\frac{R}{T} \ln K$	Σ
820	1,093	0.915	0.0241	0.000933	0.0175	$1.524 \cdot 10^{-8}$	75.793	-85.303
850	1,123	.8905	.0517	.00221	.0374	$1.826 \cdot 10^{-7}$	30.855	-90.621
890	1,163	.860	.1159	.00521	.0836	$2.269 \cdot 10^{-6}$	25.845	-96.122
900	1,173	.8525	.1395	.00645	.1006	$4.186 \cdot 10^{-6}$	24.626	-97.465
920	1,193	.8380	.2132	.01026	.1535	$1.615 \cdot 10^{-5}$	21.941	-100.395
930	1,203	.8315	.2278	.01054	.1641	$1.822 \cdot 10^{-5}$	21.703	-100.755
940	1,213	.8245	.2843	.01355	.2050	$3.762 \cdot 10^{-5}$	20.259	-102.321
950	1,223	.8180	.3226	.01514	.2323	$5.32 \cdot 10^{-5}$	19.570	-103.131
960	1,233	.8110	.4081	.02040	.2931	$1.221 \cdot 10^{-4}$	17.916	-104.898
970	1,243	.8045	.4474	.02146	.3220	$1.483 \cdot 10^{-4}$	17.532	-105.405
980	1,253	.7980	.5093	.02422	.3667	$2.157 \cdot 10^{-4}$	16.786	-106.262
990	1,263	.7920	.5978	.02854	.4302	$3.480 \cdot 10^{-4}$	15.835	-107.331
1,000	1,273	.7855	.6935	.03330	.4995	$5.54 \cdot 10^{-4}$	14.911	-108.376
1,020	1,293	.7733	.887	.04223	.639	$1.142 \cdot 10^{-3}$	13.472	-110.043
1,030	1,303	.7677	1.024	.04898	.738	$1.770 \cdot 10^{-3}$	12.607	-111.035
1,040	1,313	.7618	1.144	.0541	.823	$2.41 \cdot 10^{-3}$	11.987	-111.769
1,050	1,323	.7560	1.330	.0636	.957	$3.870 \cdot 10^{-3}$	11.045	-112.828
1,060	1,333	.7503	1.515	.0721	1.091	$5.67 \cdot 10^{-3}$	10.286	-113.702
1,070	1,343	.7445	1.747	.0841	1.257	$8.89 \cdot 10^{-3}$	9.391	-114.711
1,080	1,353	.7390	1.931	.0924	1.391	$1.188 \cdot 10^{-2}$	8.814	-115.404
1,090	1,363	.7338	2.297	.1123	1.654	$2.087 \cdot 10^{-2}$	7.696	-116.629
1,100	1,373	.7282	2.632	.1301	1.893	$3.209 \cdot 10^{-2}$	6.841	-117.606
1,110	1,383	.7230	3.620	.1475	2.130	$4.634 \cdot 10^{-2}$	6.108	-118.451

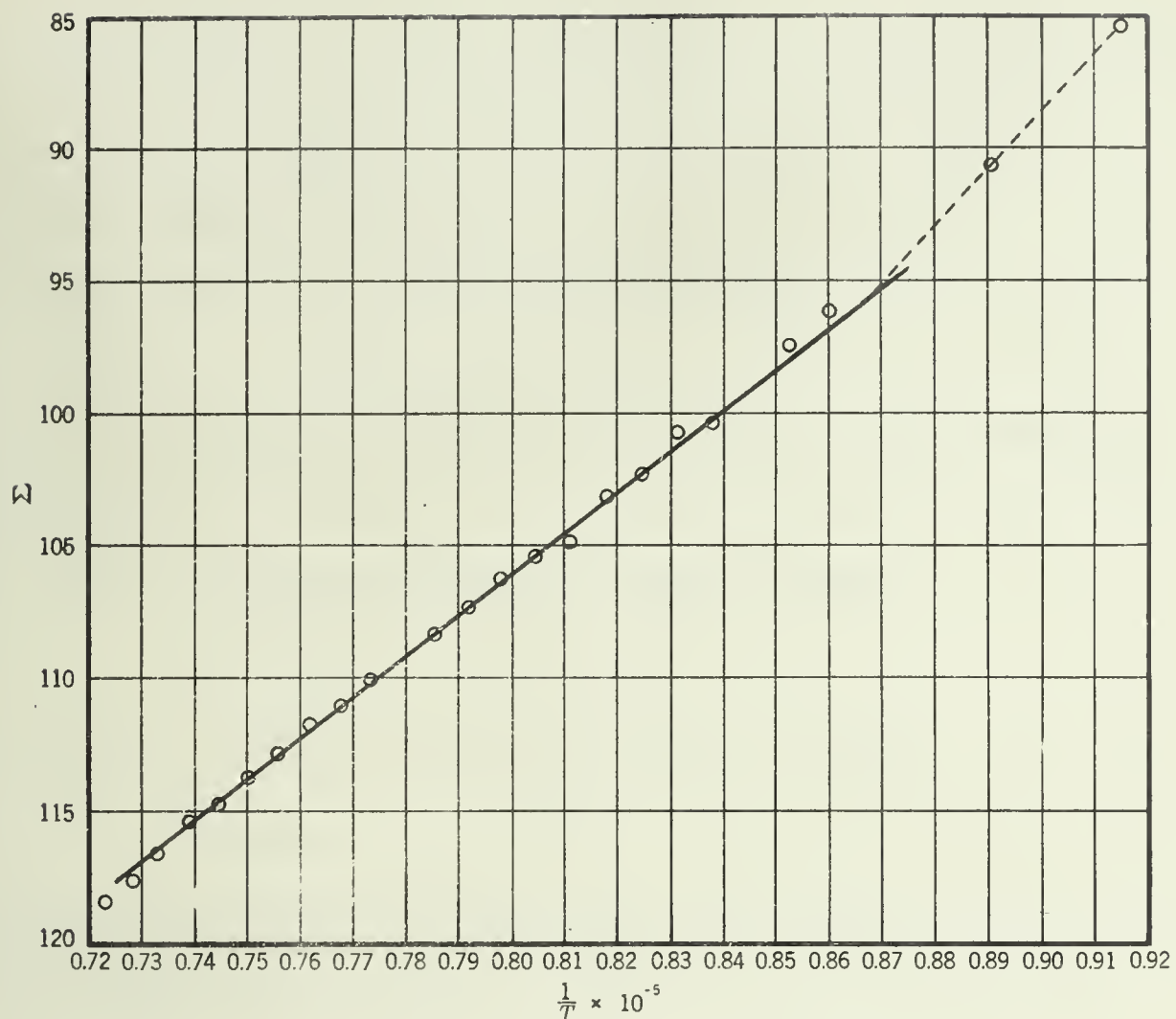


Figure 11.- Sigma function for the reaction $3\text{VnSO}_4 = \text{Mn}_3\text{O}_4 + 2\text{SO}_2 + \text{SO}_2$.

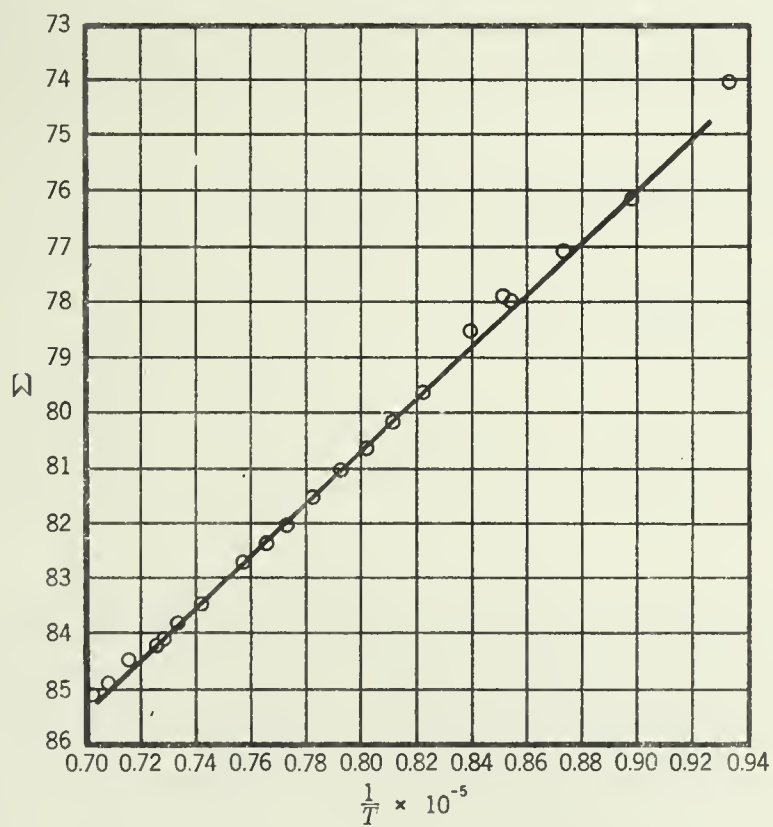
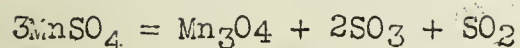


Figure 12.- Sigma function for the vaporization of MnCl_2 .



Thus, for the reaction



$$\Delta H_T = 153,800 - 18.65 T + 1.285 \cdot 10^{-2} T^2 - 0.78 \cdot 10^{-5} T^3$$

$$\Delta H_{298} = +149,186,$$

$$\Delta F_T = 153,800 + 18.65 T \ln T - 1.285 \cdot 10^{-2} T^2 + 0.39 \cdot 10^{-5} T^3 - 229.08 T,$$

$$\Delta F_{298} = +117,210.$$

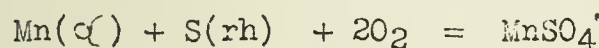
$$\text{For MnSO}_4, \Delta H_{298} = \frac{-345,000 + 2(93,540) + 70,940 + 149,186}{3} \\ = -250,735$$

$$\Delta F_{298} = \frac{-519,820 + 2(86,670) + 70,440 + 117,210}{3} \\ = -226,935$$

The figures for the heat and free energy of SO_2 above are taken from the recently recalculated figures of Eastman (79) and the figures for SO_3 are obtained by correcting Lewis and Randall's calculation for this material in accordance with Eastman's SO_2 revision.

Berthelot's figure for MnSO_4 was -249,400, which is to be compared to the calculated one of -250,735. The difference of approximately 0.4 percent is perhaps all that could be expected for this sort of calculation. The latter figure will be used for further calculations.

As final check, the entropy of MnSO_4 may be calculated. For the reaction



$$\Delta S = \frac{-250,735 - 226,935}{298} = -79.8$$

$$S_{298}(\text{MnSO}_4) = \underline{x} = 33.1.$$

In the absence of any direct data, the result 33.1 for the entropy of MnSO_4 seems a reasonable figure.

It is evident that the slope of the dotted line will give a hypothetical value for MnSO_4 differing by approximately 23,000 calories from that directly determined. This difference is too great to be accounted for either by fusion or by a transition in the solid phase. It must be concluded that the upper points are in error or that a different reaction is taking place here. It seems well-proved that Mn_3O_4 is the actual product of decomposition, so that the formation of a basic salt is improbable. It can be shown that the oxygen-dissociation pressure of Mn_2O_3 at these temperatures is considerably greater than that

from SO_3 , so that the reaction here could not be the formation of this oxide. The formation of MnO_2 is even more improbable. The heat of reaction $\text{MnSO}_4 = \text{MnO} + \text{SO}_3$ is very far from the observed slope. The direction of the difference between the dotted line and the full line is such that a larger dissociation pressure would bring better concordance; and in view of this fact, the author is inclined to assume that equilibrium was not attained at the lower temperatures. These arguments may lack conviction, and it is certain that further work in this interesting problem should be done.

MANGANOUS CHLORIDE

Manganous chloride, like the sulphate, has a number of hydrated crystal forms which have little interest for the present purpose. The anhydrous material, which has been found naturally as a product of volcanic activity, has a density of approximately 3.0, and according to Sandonnini (335) melts at 650°C . The vapor density is normal.

Specific Heats

No specific heats of MnCl_2 at low temperatures are available, and the only determinations on this material seem to be those of Regnault (312), who reported 0.14335 for the mean specific heat per gram from 98.90 to 22.22° , and 0.14175 from 98.74 to 22.50°C . The figure 18.05 per formula weight at 60°C . will be taken as representing these results. As before, the specific heat per gram-atom will be taken as 7.0 at the melting point.

$$\text{Then } 18.05 = \underline{a} + 60 \underline{b},$$

$$21.0 = \underline{a} + 650 \underline{b},$$

$$\underline{b} = 5 \cdot 10^{-3},$$

$$\underline{a} = 17.75,$$

$$\underline{C}_p(\text{MnCl}_2(\text{s})) = 17.75 + 5 \cdot 10^{-3} \underline{t},$$

$$= 16.38 + 5 \cdot 10^{-3} \underline{T}.$$

For liquid MnCl_2 , it seems probable that the specific heat should be approximately 8 per gram atom, that is, $\underline{C}_p(\text{MnCl}_2(\text{l})) = 24.0$. Consideration of the possible degrees of freedom of the atoms in the vapor led to the theoretical value 12. Perhaps the expression $\underline{C}_p(\text{MnCl}_2(\text{g})) = 12 + 2 \cdot 10^{-3} \underline{T}$ will not be greatly in error.

Vapor Pressure

The only available vapor-pressure measurements on MnCl_2 are those of the present author (237). It seems worth while to set up a Σ -function plot for

these results. Thus, in the reaction



$$\underline{C}_p(\text{MnCl}_2(g)) = 12.0 + 2 \cdot 10^{-3} \underline{T}$$

$$\underline{C}_p(\text{MnCl}_2(l)) = 24.$$

$$\Delta \underline{C}_p = -12 + 2 \cdot 10^{-3} \underline{T}$$

$$\Sigma = -R \ln p - 12 \ln T + 1 \cdot 10^{-3} T.$$

The various individual figures for the Σ plot are shown in table 53, the corresponding chart in figure 12. The fact that the use of the specific heats selected above largely eliminates the curvature of the Σ v. $\frac{1}{T}$ plot compared with the $\log p$ v. $\frac{1}{T}$ curve given in the original publication confirms the order of magnitude of these assumptions. The points in the central part of the chart show an excellent conformance to the theoretical straight line, and deviations occur only in the high and low ranges, where the accuracy of the experimental method was less good.

TABLE 53. - Σ Function for Vaporization of MnCl_2

Temperature, °K.	$\frac{1}{T} \cdot 10^3$	V.P.		Log V.P., atmospheres	$-R \ln p$	Σ
		Millimeters	Atmospheres			
1,071.7	0.9333	10.0	0.0131	-1.8808	8.612	-74.043
1,113.5	.8980	25.1	.0330	-1.4813	6.782	-76.290
1,145.6	.8733	33.0	.0434	-1.3624	6.238	-77.147
1,160.3	.8518	41.1	.0541	-1.2670	5.801	-77.722
1,157.1	.8542	47.2	.0621	-1.2070	5.526	-77.956
1,190.6	.8398	54.3	.0714	-1.1461	5.249	-78.553
1,215.3	.8227	83.7	.1101	-.9584	4.388	-79.630
1,232.2	.8115	102.4	.1346	-.8706	3.986	-80.184
1,246.1	.8023	121.0	.1593	-.7980	3.654	-80.637
1,261.4	.7928	138.9	.1826	-.7385	3.381	-81.042
1,278.1	.7828	163.0	.2210	-.6556	3.002	-81.547
1,293.3	.7732	201.0	.2644	-.5778	2.646	-82.029
1,305.9	.7657	226.4	.2980	-.5259	2.408	-82.384
1,320.5	.7572	251.2	.3305	-.4808	2.201	-82.714
1,347.7	.7420	330.9	.4352	-.3614	1.655	-83.460
1,363.2	.7338	374.8	.4930	-.3071	1.406	-83.832
1,372.6	.7283	410.0	.5395	-.2681	1.228	-84.094
1,378.5	.7255	426.1	.5605	-.2514	1.151	-84.204
1,398.6	.7152	455.0	.5985	-.2228	1.020	-84.489
1,412.8	.7080	529.1	.696	-.1575	.721	-84.904
1,423.3	.7025	565.5	.744	-.1288	.590	-85.110

The slope of the Σ -function plot indicates that the heat of vaporization is

$$\underline{H}_0 = +46,950,$$

$$\underline{H}_T = 46,950 - 12 T - 2 \cdot 10^{-3} T^2,$$

$$\underline{H}_{298} = +43,550,$$

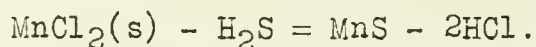
$$\underline{F}_T^0 = 46,950 - 12 T \ln T - 1 \cdot 10^{-3} T^2 - 118.27 T,$$

$$\underline{F}_{298}^0 = +31,972.$$

The free-energy equation above is undoubtedly better for purposes of extrapolation than the original $\log p$ v. $\frac{1}{T}$ plot in the original reference. In view of this fact, table 54 and figure 13 show the vapor pressures calculated from the equation. Using this method of extrapolation, the normal boiling point is $1,175^\circ\text{C}$., whereas the figure $1,190^\circ$ was obtained by the less-accurate method of extrapolation used previously.

Thermal and Equilibrium Data

Using a dynamic method, Jellinek and Von Podjaski (179) have determined the equilibrium concentration of the gases for the reaction



at three different temperatures, approaching equilibrium from both sides at the intermediate point. These figures may be used to calculate the free energy of the solid chloride. Thus

$$\underline{C}_p(2\text{HCl}) = 13.52 + 1.212 \cdot 10^{-3} T + 0.26 \cdot 10^{-6} T^2$$

$$\underline{C}_p(\text{MnS}) = 10.21 + 6.56 \cdot 10^{-3} T - 2.42 \cdot 10^{-6} T^2$$

$$\text{Sum} \quad 23.73 + 7.77 \cdot 10^{-3} T - 2.16 \cdot 10^{-6} T^2$$

$$\underline{C}_p(\text{H}_2\text{S}) = 7.20 + 3.60 \cdot 10^{-3} T$$

$$\underline{C}_p(\text{MnCl}_2) = 16.38 + 5.0 \cdot 10^{-3} T$$

$$\text{Sum} \quad 23.58 + 8.60 \cdot 10^{-3} T$$

$$\text{Difference} = \Delta \underline{C}_p = 0.15 - 0.83 \cdot 10^{-3} T - 2.16 \cdot 10^{-6} T^2$$

$$\Sigma = -R \ln K + 0.15 \ln T - 0.415 \cdot 10^{-3} T - 0.36 \cdot 10^{-6} T^2.$$

The various items calculated for this latter formula are listed in table 55 and shown graphically in figure 14 as Σ v. $\frac{1}{T}$.

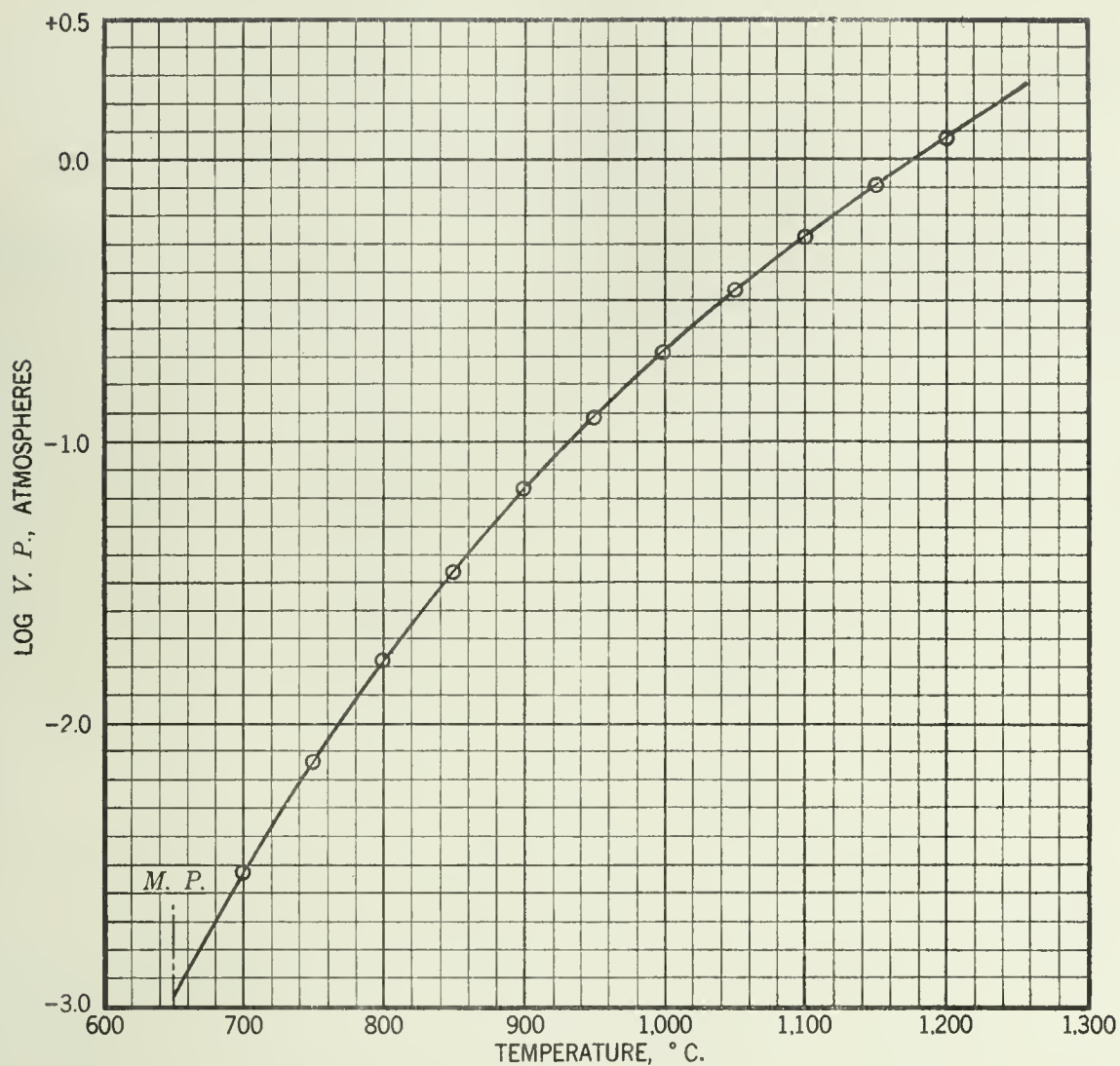


Figure 13.- Vapor pressure of MnCl₂.

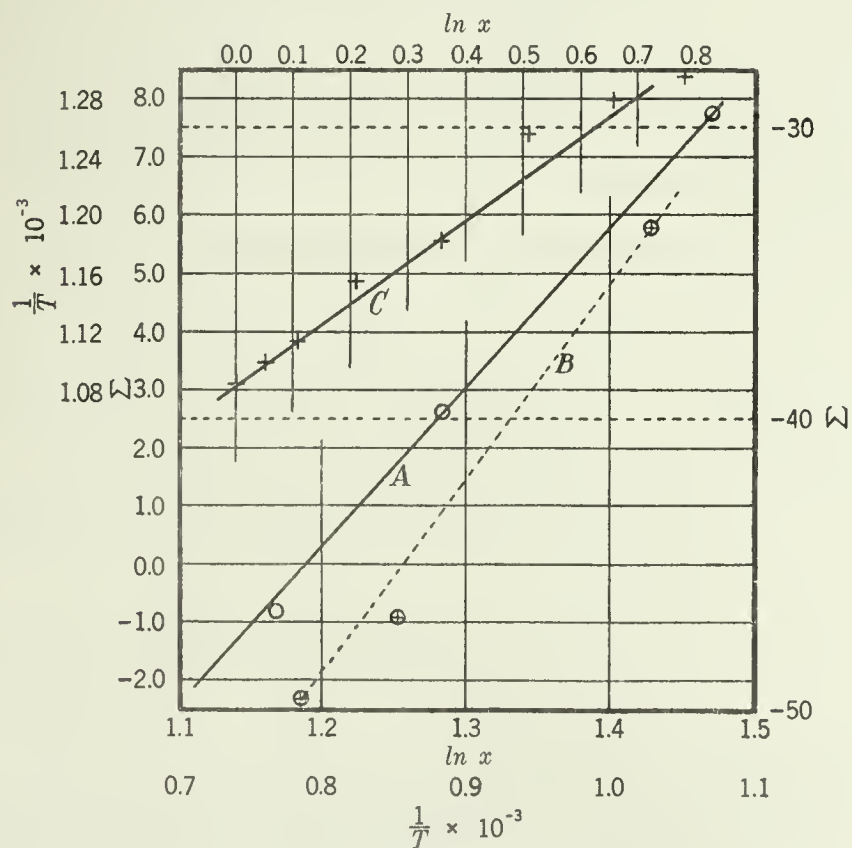


Figure 14.- Equilibrium data for MnCl₂: A, MnCl₂(s) + H₂S = MnS + 2HCl; B, MnCl₂(l) + H₂ = Mn(B) + 2HCl; C, MnCl₂(s) = MnCl₂(l).

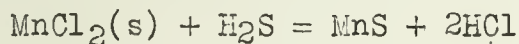
TABLE 54. - Vapor Pressures of MnCl_2

Temperature, °C.	Log V.P., atmospheres	V.P.	
		Atmospheres	Millimeters
650.....	-2.969	$1.08 \cdot 10^{-3}$	0.8
700.....	-2.526	$2.98 \cdot 10^{-3}$	2.3
750.....	-2.133	$7.36 \cdot 10^{-3}$	5.6
800.....	-1.779	$1.67 \cdot 10^{-2}$	12.7
850.....	-1.462	$3.46 \cdot 10^{-2}$	26.3
900.....	-1.170	$6.76 \cdot 10^{-2}$	51.3
950.....	-.917	.1209	91.9
1,000.....	-.683	.2072	157.5
1,050.....	-.468	.3456	262.6
1,100.....	-.272	.5355	407.0
1,150.....	-.092	.8097	615.5
1,175.....	0	1.0	760.
1,200.....	+.073	1.182	898.

TABLE 55. - Σ Function for Reaction $\text{MnCl}_2(\text{s}) + \text{H}_2\text{S} = \text{MnS} + 2\text{HCl}$

Temperature,		$\frac{1}{T} \text{ } ^\circ\text{K} \cdot 10^3$	$K = \frac{\text{PHCl}_2}{\text{pH}_2\text{S}}$	$-R \ln K$	Σ
°C.	°K.				
407	680	1.471	0.0265	+7.220	+7.751
506	779	1.284	.335	+2.175	+2.634
583	856	1.168	1.852	-1.226	-.831

The heat of the equilibrium reaction may be determined from thermal data:



$$\Delta H_{298} -112,600 -4,760 -45,890 \quad 2(-22,000) \quad \Delta H = +27,470$$

The heats for H_2S and HCl are those used by Lewis and Randall, that for MnCl_2 is Berthelot's, and the MnS figure is the one selected in an earlier part of this work.

$$\text{Then } \Delta H_{298} = +27,470 = \Delta H_0 + 0.15 T - 0.415 \cdot 10^{-3} T^2 - 0.72 \cdot 10^{-6} T^3,$$

$$\Delta H_0 = +27,481.$$

The middle point of the data of Jellinek and Podjaski may be selected as the most reliable, because it was approached from both sides of the equilibrium. The line shown in the chart is drawn through this point with the theoretical

slope of $\Delta H_o = +27,481$. The line so drawn is almost precisely the one that might have been put through the points shown without reference to other data, and a very satisfactory interpretation of the data is thus apparent.

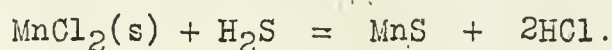
The heats and free energy of the reaction are then given by the expressions

$$\Delta H_T = +27,481 + 0.15 T - 0.415 \cdot 10^{-3} T^2 - 0.72 \cdot 10^{-6} T^3,$$

$$\Delta F_T^0 = +27,481 - 0.15 T \ln T + 0.415 \cdot 10^{-3} T^2 + 0.36 \cdot 10^{-6} T^3 - 32.64 T,$$

$$\Delta F_{298}^0 = +17,538.$$

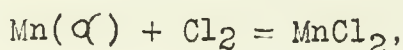
Again writing the reaction



$$\Delta F \quad \quad \quad -7,840 \quad -47,025 \quad 2(-22,692) \quad \Delta F = +17,538$$

$$\Delta F = \Delta F_{298}^0(\text{MnCl}_2(\text{s})) = -102,107.$$

Finally, the entropy may be calculated

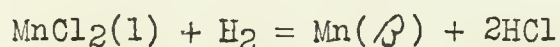


$$7.3 \quad 53.0 \quad \underline{s} \quad \Delta S = \frac{-112,600 + 102,107}{298} = -35.2$$

$$S_{298}(\text{MnCl}_2(\text{s})) = \underline{s} = 25.1.$$

Further equilibria for the reduction of MnCl_2 by H_2 , have been measured at higher temperatures by Jellinek and Rudat (178). These authors give three determinations of equilibrium concentrations, at 700° , 900° , and $1,000^\circ\text{C}$. It has been shown in an earlier part of this work that a transition in solid manganese probably occurs at 835°C ., involving 148 calories per Mn. This small heat change is well within the uncertainty of our knowledge of the heats of formation of MnCl_2 , and the results will be first calculated on the basis of β manganese.

The reaction on this basis is



$$C_p(2\text{HCl}) = 13 + 2 \cdot 10^{-3} T$$

$$C_p(\text{Mn}\beta) = \underline{9}$$

$$\text{Sum} \quad \quad \quad 22.00 + 2 \cdot 10^{-3} T$$

$$C_p(\text{H}_2) = 6.50 + 0.9 \cdot 10^{-3} T$$

$$C_p(\text{MnCl}_2(\text{l})) = \underline{24.0}$$

$$\text{Sum} \quad \quad \quad 30.50 + 0.9 \cdot 10^{-3} T$$

$$\text{Difference} = \Delta C_p = -8.50 + 1.1 \cdot 10^{-3} T$$

$$\sum = -R \ln K - 8.50 \ln T + 0.55 \cdot 10^{-3} T.$$

TABLE 56. - Σ Function for Reaction $\text{MnCl}_2(1) + \text{H}_2 = \text{Mn}(2) + 2\text{HCl}$

Temperature oC.	oK.	$\frac{1}{T} \cdot 10^3$	HCl, percent	Press- ure, mm.	V.P. MnCl ₂ , mm.	$\frac{p_{\text{HCl}}}{\text{atm}}$	$\frac{p_{\text{H}_2}}{\text{atm}}$	$\frac{K = \frac{p_{\text{HCl}}^2}{p_{\text{H}_2}}}{\text{atm}}$	$-R \ln \frac{K}{T}$	$-8.50 \ln \frac{T}{T}$	$0.55 \cdot 10^{-3} T$	Σ
700	973	1.028	0.21	760	2.3	$2.09 \cdot 10^{-3}$	0.995	$4.402 \cdot 10^{-6}$	24.525	-58.481	+0.535	-33.421
900	1,173	.8525	4.28	773	51.3	$4.06 \cdot 10^{-2}$.909	$1.815 \cdot 10^{-3}$	12.550	-60.067	.644	-46.873
1,000	1,273	.7857	7.85	773	157.5	$6.36 \cdot 10^{-2}$.746	$5.415 \cdot 10^{-3}$	10.379	-60.771	.700	-49.692

The individual figures for the various components of Σ are shown in table 56 and plotted in figure 14. The values 4.28 and 7.85 differ from those given by the original authors (4.4 and 7.7), because in replotting their individual analysis at various rates of gas flow, and extrapolating to zero flow, greater weight has been given to the figures at higher velocity. There seems good reason for doing so, since at the upper temperatures the vapor pressure of MnCl_2 is considerable, and experimentally there seemed to be considerable change for reversal of the equilibrium in the exit parts of the apparatus.

The points on the Σ -function plot so obtained do not lie upon a straight line, nor is there available direct evidence for the slope of the curve. The proper line may be determined if the heat of fusion of MnCl_2 is known. Fortunately, the heat of fusion of MnCl_2 may be calculated from the melting points of the MnCl_2 - SrCl_2 system investigated by Sandonnini (336). These two materials form a nearly perfect solution, with a well-defined eutectic.

For a perfect solution, the heat of fusion may be obtained from the expression

$$\begin{aligned} R \ln x &= \frac{L(T_{F0} - T_F)}{T_{F0} T_F} \\ &= \frac{L}{T_F} - \frac{L}{T_{F0}}, \end{aligned}$$

where x is the mole fraction of the constituent whose pure crystals are in equilibrium with the melt, L the heat of fusion, T_{F0} and T_F the freezing points of the pure constituent and melt, respectively. Further, if $\ln x$ is plotted against $\frac{1}{T_F}$, the slope of the line is $\frac{L}{R}$.

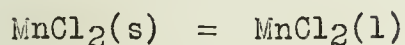
Table 57 shows the individual figures corresponding to the experimental points of Sandonnini and figure 14 is a chart of the points as indicated above.

TABLE 57. - Melting-Point Data for System MnCl_2 - SrCl_2

Mole, percent MnCl_2	$\ln x$	T_F	$\frac{1}{T_F} \cdot 10^3$
100	0	923	1.084
95	0.0509	910	1.099
90	.1060	899	1.113
80	.2089	866	1.155
70	.357	846	1.182
60	.509	796	1.256
50	.693	782	1.279
46	.776	772	1.295

Reference to the chart shows that the points in the table may fairly well be represented by a straight line with a slope corresponding to 7,110 calories.

Then for the reaction



$$\underline{C}_p(\text{MnCl}_2(\text{l})) = 24.0$$

$$\underline{C}_p(\text{MnCl}_2(\text{s})) = \underline{16.38 + 5.0 \cdot 10^{-3} \underline{T}}$$

$$\Delta \underline{C}_p = 7.62 - 5.0 \cdot 10^{-3} \underline{T}.$$

Furthermore,

$$\Delta \underline{H}_{923} = 7,110 = \Delta \underline{H}_0 + 7.62 \underline{T} - 2.5 \cdot 10^{-3} \underline{T}^2,$$

$$\Delta \underline{H}_0 = 2,206,$$

$$\Delta \underline{H}_{\underline{T}} = 2,206 + 7.62 \underline{T} - 2.5 \cdot 10^{-3} \underline{T}^2,$$

$$\Delta \underline{H}_{298} = +4,256.$$

Similarly,

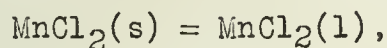
$$\Delta \underline{F}_{923} = 0 = 2,206 - 7.62 \underline{T} \ln \underline{T} + 2.5 \cdot 10^{-3} \underline{T}^2 + \underline{I} \underline{T},$$

$$\underline{I} = +47.30,$$

$$\Delta \underline{F}^0_{\underline{T}} = 2,206 - 7.62 \underline{T} \ln \underline{T} + 2.5 \cdot 10^{-3} \underline{T}^2 + 47.30 \underline{T},$$

$$\Delta \underline{F}^0_{298} = +3,603.$$

Referring again to the melting reaction



$$\Delta \underline{H}_{298} \quad -112,600 \quad \underline{x} \quad \Delta \underline{H} = +4,256$$

$$\underline{x} = \Delta \underline{H}_{298}(\text{MnCl}_2(\text{l})) = -108,344,$$

$$\Delta \underline{F}_{298} \quad -102,107 \quad \underline{y} \quad \Delta \underline{F} = +3,603$$

$$\underline{y} = \Delta \underline{F}^0_{298}(\text{MnCl}_2(\text{l})) = -98,504.$$

Since the reduction reaction has been written to involve β manganese, the free energy and heat of this form are also needed.

In the reaction



$$C_p(\text{Mn}\beta) = 9.00,$$

$$C_p(\text{Mn}\alpha) = 3.78 + 7.38 \cdot 10^{-3} T,$$

$$\Delta C_p = 5.22 - 7.38 \cdot 10^{-3} T.$$

At the transition point, $835^\circ\text{C.} = 1,108^\circ\text{K.}$,

$$\Delta H_{1,108} = 148 = \Delta H_0 + 5.22 T - 3.69 \cdot 10^{-3} T^2,$$

$$\Delta H_0 = -1,102,$$

$$\Delta H_T = -1,102 + 5.22 T - 3.69 \cdot 10^{-3} T^2,$$

$$\Delta H_{298} = +126,$$

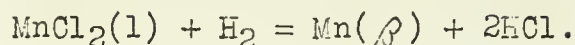
$$\Delta F_{1,108}^0 = 0 = -1,102 - 5.22 T \ln T + 3.69 \cdot 10^{-3} T^2 + I T,$$

$$I = +33.52,$$

$$\Delta F_T^0 = -1,102 - 5.22 T \ln T + 3.69 \cdot 10^{-3} T^2 + 33.52 T,$$

$$\Delta F_{298}^0 = +456.$$

Returning now to the reduction reaction:



$$\Delta H_{298} \quad -108,344 \quad 0 \quad +126 \quad 2(-22,000) \quad \Delta H = +64,470$$

$$\Delta F_{298}^0 \quad -98,504 \quad 0 \quad +456 \quad 2(-22,692) \quad \Delta F = +53,576$$

$$\Delta H_{298} = 64,470 = \Delta H_0 - 8.50 T + 0.55 \cdot 10^{-3} T^2,$$

$$\Delta H_0 = +66,957.$$

The theoretical slope for the line of the Σ -function plot shown in figure 14, in accordance with these computations, should be $\Delta H_0 = +66,957$. If the upper point is assumed the most accurate one (because of the low vapor pressure of MnCl_2) a line drawn through it with this slope strikes the lowest point very closely, but missing the intermediate one. The line so drawn is represented by the free-energy equation

$$\Delta F_T^0 = +66,957 - 8.50 T \ln T - 0.55 \cdot 10^{-3} T^2 - 102.22 T,$$

the value of integration constant $I = -102.22$ being determined from the upper point. Whence, by calculation to 298°K. , there is found $\Delta F_{298}^0 = +50,880$.

This value does not compare too well with the result +53,576, calculated directly from fusion data. The discrepancy cannot be due to error in the heat of fusion of manganese chloride, because even considerable changes in the value of the selected heat, if introduced into the above estimates, largely compensate, leaving the discrepancy virtually unchanged. Moreover, equilibrium and thermal data are so well in accord for the previously calculated reaction between the chloride and hydrogen sulphide, that error in the solid MnCl_2 results hardly seems probable. It seems more probable to suppose that the relatively great vapor pressure of MnCl_2 caused experimental errors (perhaps due to reversal of the reaction on cooling the gaseous products). On this basis, the figures for liquid MnCl_2 will be taken from those calculated from the solid, and the melting reaction.

The measurements of the decomposition voltage of MnCl_2 by Devoto and Guzzi (69) are obviously unsuited to calculation of free energy of fused MnCl_2 , as the authors themselves showed from related data.

MANGANESE CARBIDE

In spite of the technical importance of manganese carbide, there are few investigations on record furnishing thermodynamically useful data. According to Stadeler (353), carbon forms with manganese the compound Mn_3C , melting at $1,217^\circ\text{C}$. and with a density of 6.9. Stadeler's melting-point determinations show mutual solubility, with a maximum in the melting curve at 3 percent carbon ($\text{Mn}_3\text{C} = 6.8$ percent) and at $1,270^\circ\text{C}$. The correct melting point of pure manganese has been taken in this work at $1,220^\circ\text{C}$., whereas Stadeler gave $1,207$.

Other melting points for this composition of carbide are given by Ruff and Gersten (329), who found $1,243^\circ$ as the mean of three determinations, and by Heraeus (148) at $1,245^\circ\text{C}$.

The manganese metal used by Stadeler analyzed only 95.8 percent manganese, and contained more than 1 percent each iron, aluminum, and silicon. The finality of measurements on so impure a material may be questioned, and it is not impossible that the melting diagram carried out with pure manganese might not show the maximum observed. It is probable, however, that the mixed crystals revealed by Stadeler's work are well-authenticated.

The carburization of manganese with methane was studied by Hilpert and Paunescu (163). Upon passing a mixture of equal parts of methane and hydrogen over electrolytically prepared manganese for long periods of time (45 hours), these investigators found the following limiting values of carbon content were reached: 600°C ., 5.7 percent C; 700° , 8.6 percent C; 800° , 13.5 percent C; 900° , 15 percent C. One must conclude from these experiments either that carbides of higher carbon content than Mn_3C are capable of forming or that the analyses made included elementary carbon from the thermal decomposition of methane. The authors took precautions, however, to remove elementary carbon, and the former alternative seems preferable. It will be necessary to return to these data later.

The solubility of carbon in liquid manganese has been measured by Ruff and Bormann (328), who found 6.82 percent C at 1,312°, 6.93 percent at 1,427°, 7.05 percent at 1,492°, and 7.12 percent at 1,525°. These authors also determined that the latter solution boiled under 30 mm pressure at 1,526°C., and upon heating further they determined that the vapor contained 1.94 percent C, the maximum temperature used being near 2,000° - 2,100°C. Pure manganese boils at 30 mm pressure at 1,511°C., and the low percentage of carbon in the vapor shows that the gas is mostly uncombined manganese. Then the liquid must have a relatively high concentration of free manganese. If the vapor pressure of the carbide at 1,526° is neglected, one may calculate the mole fraction of manganese in the liquid from Raoult's law. By interpolation in table 46 (p.106), the vapor pressure of pure manganese at 1,526°C. is 33.35 mm, whence the mole fraction of manganese is $\frac{30}{33.35} = 0.899$, yet the melt contains slightly more than enough carbon to convert all the manganese to Mn_3C and is saturated with carbon, since the experiments were performed in graphite crucibles.

It is not allowable, however, to assume that the carbon is present in solution as such, for then the mole fraction of manganese would be

$$\left(\frac{100 - 7.12}{55}\right) \div \left(\frac{100 - 7.12}{55} + \frac{7.12}{12}\right) = 0.74,$$

and a much greater raising of the boiling point would ensue. The accuracy of Ruff and Bormann's boiling-point determinations is well attested by the calculations from spectroscopic data made in an early part of this work, and one must assume that the figure 0.899 is quite precise.

In seeking explanation for this anomalous condition, one may assume other carbides, or mixtures of carbides and carbon, to be present. Thus assuming two unknown proportions, say x and y , of any two constituents, simultaneous expressions may be set up, one showing the mole fraction of manganese and the other the total carbon content, both of these quantities being known and fixed. The solution of these equations for Mn_2C and Mn_3C , Mn_2C and C, MnC and Mn_3C , MnC and C, MnC_2 and Mn_3C , MnC_2 and C, all lead to impossible values, either x or y being negative. It is, moreover, apparent from the quantities involved that the carbide must have a relatively high molecular weight to permit a possible and necessary 0.899 mole fraction of pure manganese to exist.

By analogy with the oxide, one may finally try Mn_3C_4 as the molecular species involved. The ratio

$$\frac{Mn_3C_4}{4C} = \frac{212.8}{48} = 4.429,$$

then 7.12 percent carbon corresponds to $7.12 \times 4.429 = 31.55$ percent Mn_3C_4 , and the mole fraction of manganese is

$$\left(\frac{100 - 31.55}{55}\right) \div \left(\frac{100 - 31.55}{55} + \frac{31.55}{212.8}\right) = 0.894.$$

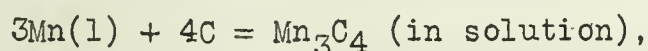
This is too close concordance with the 0.899 calculated from Raoult's law to be a mere coincidence. The conclusion seems justified, then, that the carbide in solution is Mn_3C_4 .

In the vapor phase, however, it seems doubtful whether such a complex molecule as Mn_3C_4 can be used to explain the volatility of carbon. The carbide in the vapor state is more probably Mn_3C . Thus, if one calculates the mole fraction of manganese in the vapor state (1.94 percent C) there is obtained:

$$\text{Mole fraction Manganese} = \left(\frac{100 - \frac{1.94}{6.78} \cdot 100}{55} \right) + \left(\frac{100 - \frac{1.94}{6.78} \cdot 100 + \frac{1.94}{6.78} \cdot 100}{55 + \frac{1.94}{6.78} \cdot 100} \right) = 0.899$$

Here again the concordance with the previously calculated figures is gratifying. The reason why the mole fraction so calculated from vapor formed over a 500° temperature range is so close to that calculated for the liquid will be apparent from subsequent calculations.

If the above reasoning is accepted, the solubility of carbon measured by Ruff and Bormann represents the equilibrium



and the equilibrium constant is

$$\underline{K} = \frac{A_{\text{Mn}_3\text{C}_4}}{A_{\text{C}}^4 A_{\text{Mn}}^3},$$

and the activity of Mn_3C_4 and liquid Mn may be taken as the mole fraction of each. Since the experiments were performed in contact with graphite as a solid phase, the activity of the carbon is unity and

$$\underline{K} = \frac{\text{m.f.}_{\text{Mn}_3\text{C}_4}}{\text{m.f.}_{\text{Mn}}^3}.$$

No specific heats for the above reaction are available, and one can only assume that $\Delta C_p = 0$, and consequently chart $\log \underline{K}$ v. $\frac{1}{T}$. The data for such calculation and plot are shown in table 58, and the points themselves in figure 15.

TABLE 58. - Solubility of C in Mn(l)

Temperature,		$\frac{1}{T} \cdot 10^3$	C, percent	m.f.		\underline{K}	$\log \underline{K}$
°C.	°K.			Mn_3C_4	Mn		
1,312	1,585	0.631	6.82	0.1009	0.8991	0.1388	-0.8609
1,427	1,700	.588	6.93	.1026	.8974	.1420	-.8479
1,492	1,765	.567	7.05	.1050	.8950	.1465	-.8342
1,525	1,798	.556	7.12	.1064	.8936	.1491	-.8265

It is apparent that the mole fraction of manganese changes very slowly with the temperature, and this may account for the possibility of calculating this quantity from analysis of the vapor, as was done previously.

All but the point at lowest temperature correspond to a satisfactory straight line with slope of $\Delta H = +3,090$ calories.

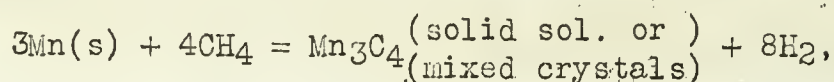
$$\text{Further, } \Delta F_{1,798} = -R T \ln K = -4.578 \cdot 1,798 \cdot (-0.8265), \\ = +6,800.$$

Then, for the reaction above,

$$\Delta F_T^0 = +3,090 + 2.062 T,$$

$$\Delta F_{298}^0 = 3,705.$$

Returning now to the experiments of Hilpert and Paunescu, the assumption may be made without inconsistency that the reaction occurring was



for which the formal equilibrium constant is

$$K = \frac{A_{\text{Mn}_3\text{C}_4}}{A_{\text{Mn}}^3} \cdot \frac{A_{\text{H}_2}^8}{A_{\text{CH}_4}^4}.$$

Here again the further assumption may be made that the activities of Mn_3C_4 and Mn are given by the respective mole fractions. Experimentally the pressures of hydrogen and methane are fixed by the equal volume mixture used in the tests. Then

$$K = \frac{\text{m.f. Mn}_3\text{C}_4}{\text{m.f. Mn}^3} \cdot \frac{0.5^8}{0.5^4} = \frac{\text{m.f. Mn}_3\text{C}_4}{\text{m.f. Mn}^3} \cdot 0.0625.$$

Table 59 shows the figures calculated from Hilpert and Paunescu's data, and figure 15 is the plot of $\log K$ v. $\frac{1}{T}$.

TABLE 59. - Formation of Solid Manganese Carbide

Temperature,		$\frac{1}{T} \cdot 10^3$	Carbon, percent	m.f.		K	log K
°C.	°K.			Mn ₃ C ₄	Mn		
600	873	1.146	5.7	0.2523	0.7477	0.0378	-1.422
700	973	1.028	8.6	.3810	.6190	.1005	-.998
800	1,073	.932	13.5	.598	.402	.575	-.241
900	1,173	.853	15.0	.664	.336	1.095	+0.049

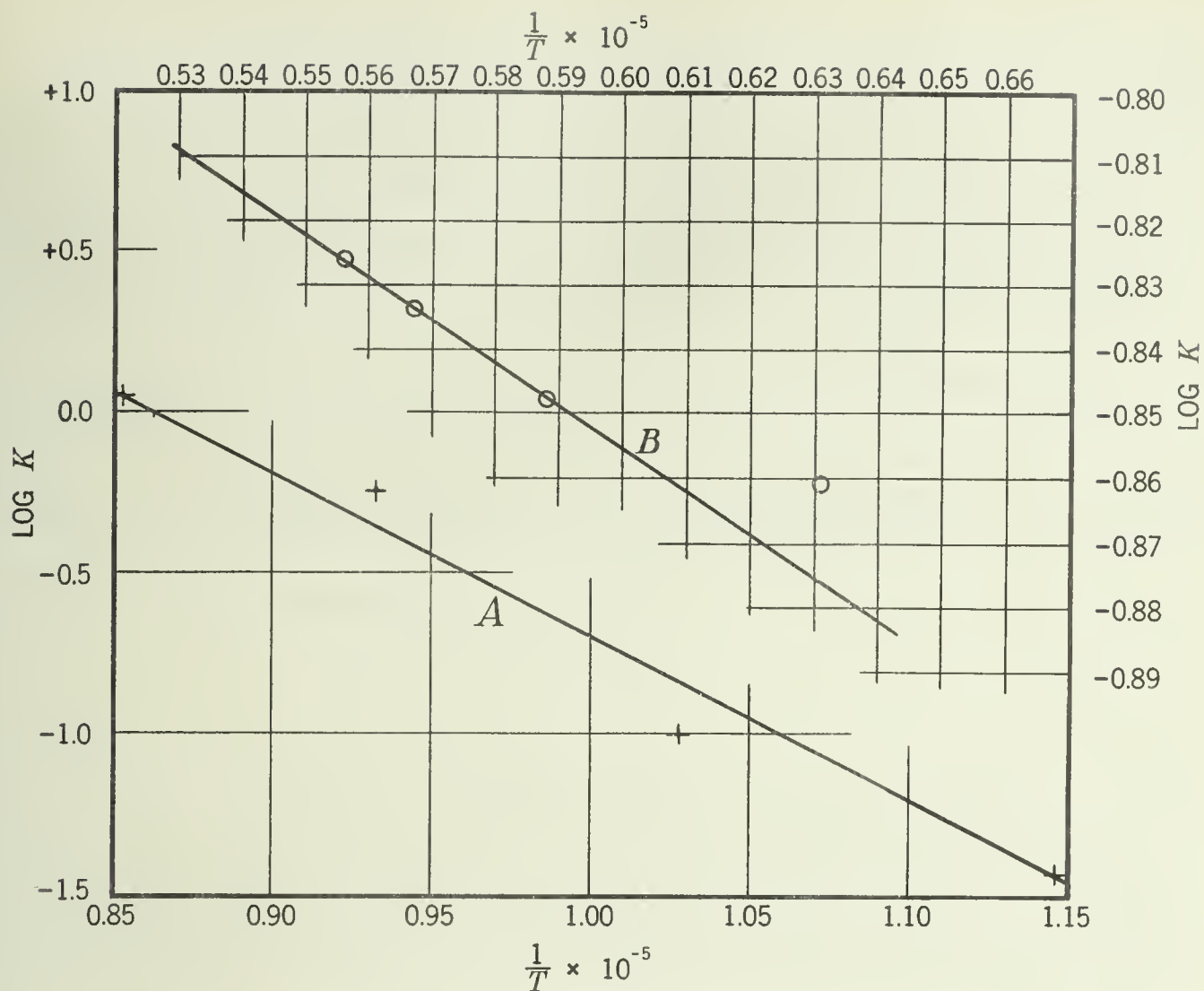


Figure 15.- Equilibria of manganese carbide: A, Solid carbide, from methane; B, liquid solution.

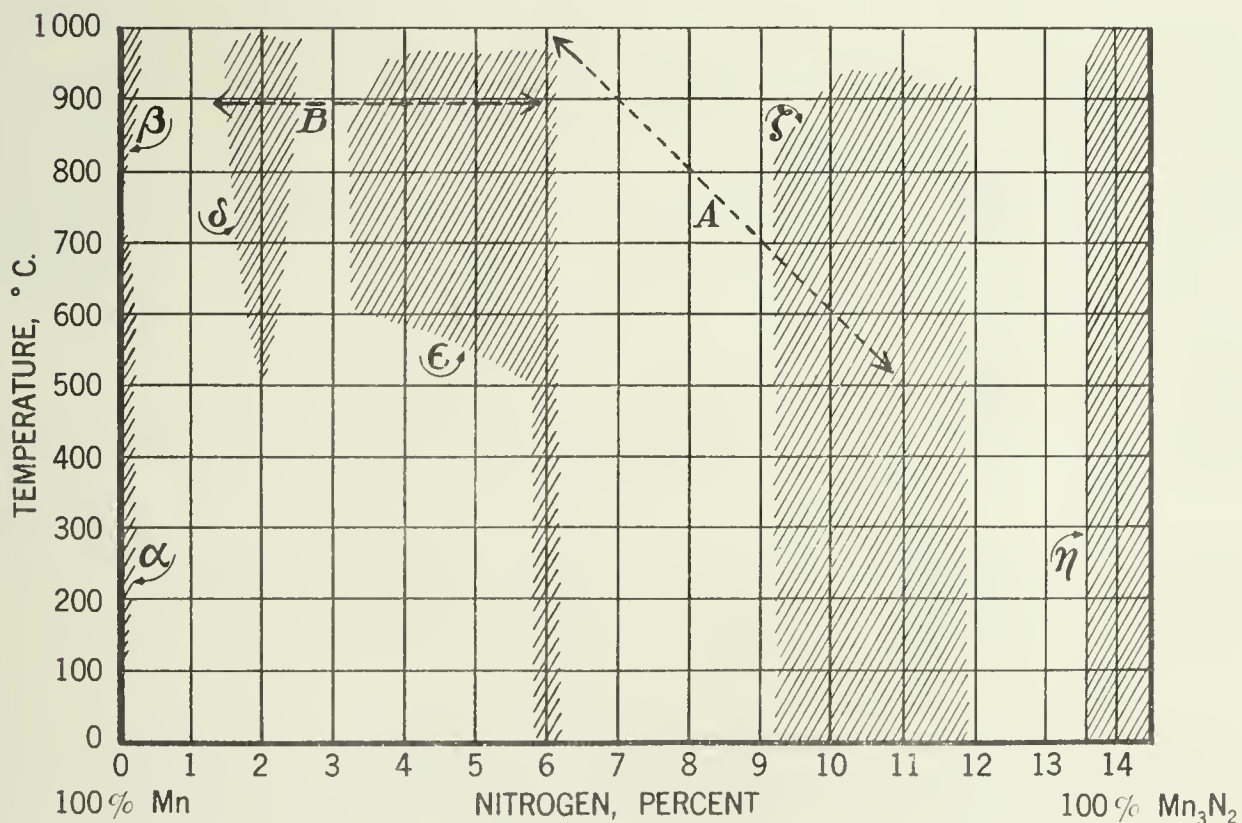


Figure 16.- Phase diagram for manganese nitrides, after Haag: A, Valenski's range; B, Tschischewsky's range.



Figure 20 shows that a fair approximation to a straight line is indicated by these points. Too great accuracy cannot be expected, since the values of final carbon content were taken from curves in the original article, and the great number of molecules involved in the formal equilibrium expression renders the calculation very sensitive to slight errors of composition of either the solid or gaseous phase.

From the slope of the line, the heat of reaction is found to be $\Delta H = +19,700$ calories.

The heat of formation of methane at ordinary temperatures is $\Delta H = -18,120$, whence the heat of formation of Mn_3C_4 in solid solution becomes

$$\Delta H = -52,780, \text{ and } \Delta F_{873} = +5,680,$$

$$\Delta F_T^0 = -52,780 + 67.00 T,$$

$$\Delta F_{298}^0 = -32,810.$$

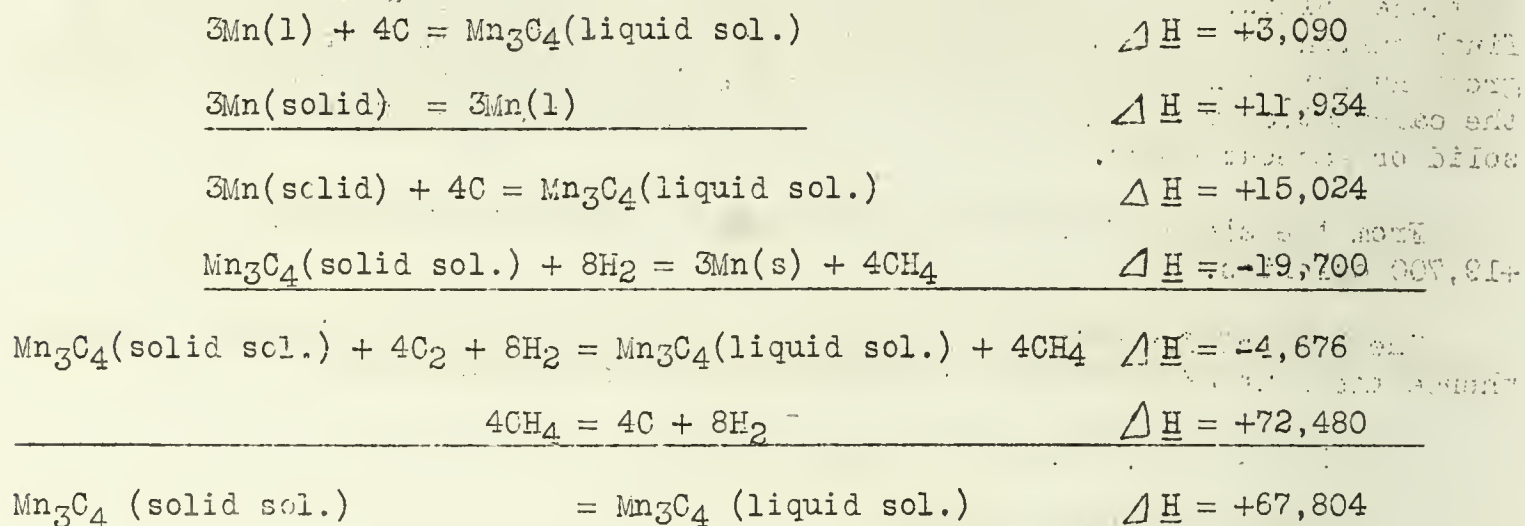
Recent direct calorimetric determinations of the heat of formation of manganese carbide are those of Roth (319) and of Ruff and Gersten (329). The former author gives $\Delta H = -23,000$ for Mn_3C , and recalculates Ruff and Gersten's data (for corresponding heat of formation of Mn_3C_4) to $\Delta H = -27,000$ calories. Roth does not give enough data in his article to permit recalculation of his results to correspond to Mn_3C_4 , but it is possible to do so with the figures of Ruff and Gersten.

These data are summarized as follows:

	No. 1	No. 2
Carbide with 6.80 percent carbon.....Grams	1.6474	1.3154
Mn_3C_4 (30.1 percent of total)..... do	.4960	.3961
Free manganese..... do	1.1514	0.9193
Heat of oxidation		
Total.....	3,782	3,070
Mn.....	2,301	1,834
Mn_3C_4	1,481	1,236
Mn_3C_4 per gram.....	2,988	3,120
Mn_3C_4 per mole.....	635,000	664,000
Heat of formation		
Mn_3O_4 (Ruff and Gersten).....	329,000	329,000
$4CO_2$	379,000	379,000
Mn_3C_4 $\Delta H =$	-63,000	-34,000

If the carbide truly is Mn_3C_4 , accurate results of calorimetric experiments on samples containing only 30 percent of the true carbide could scarcely be expected. The order of magnitude is nearly the same as that obtained from the experiments of Hilpert and Paunescu.

The calculations which have been made above seem to bring a fair degree of concordance to the available data, but in one respect they are still unsatisfactory. This is illustrated by the following sequence of reactions:



It does not seem probable that the apparent heat of fusion of Mn_3C_4 calculated in this way can be as high as indicated, but it is perhaps possible.

Obviously a more satisfactory experimental investigation of the carbides of manganese must be made before any great reliance may be placed on the thermodynamic properties. It may, however, be stated that the free-energy expressions given represent the only experimental data available.

MANGANESE NITRIDE

Elementary manganese readily absorbs gaseous nitrogen at elevated temperatures, and a number of investigations of this phenomenon have been made. Two nitrides are reported in the literature based on chemical data, Mn_5N_2 , and Mn_3N_2 , but a phase study of the systems involved leads to complicated and to some extent indefinite results, with the presumption of the compound Mn_4N . The data of Tschischewsky reported by Mellor (253) (original data not available) (254) show conclusive chemical evidence for the formation of the nitride Mn_3N_2 , when ammonia is used for nitriding between 600° and 800° C. The calculations below show that most of the quantitative data may be explained upon the assumption that only one nitride, Mn_3N_2 , is formed.

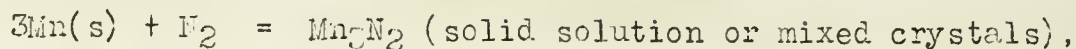
The phase changes in the Mn - N system are very important, and it is fortunate that careful X-ray studies of this system have been made by Haag (136). This author shows six distinct phases, and their disposition is best illustrated by a temperature-composition diagram, such as figure 16, which is a sketch reproduction of Haag's original.

Only for the η phase was this investigator able to assign definite positions to nitrogen and manganese atoms, and since above 500° C. this phase covers a considerable range of composition, the conclusion seems inescapable that the η phase consists of mixed crystals. The β phase, representing many of the compositions obtained by the action of nitrogen gas on metallic manganese, has, according to the author, the nitrogen atoms in den Hohlraum willkürlich eingelegen, and it may therefore be considered a solid solution, with the nitride having amorphous properties, in contrast to the crystalline η phase. Although the lower composition limit corresponds to Mn_5N_2 , it will be shown that the chemical properties are explained only by the assumption of Mn_3N_2 in lowered activity.

The most careful equilibria determinations between manganese, its nitrides, and nitrogen have been made by Valenski (377), who also reports the results of the experiments of Shukow (Zhukov, Yukoff, Jukow seem to be other references to the same work), whose original publication was inaccessible to the present author. Other authors who have also experimented in this field are Duparc, Wenger, and Cimerman (76, 77).

Although this considerable accumulation of data exists, unfortunately only part of the material is usable. Thus these last authors not only had quite impure manganese metal to work with, but also the time periods used by them lead to the presumption that equilibrium was probably not attained in the solid constituents. Thus Haag found it necessary to "homogenize" his samples by long-continued heat treatment; but some of the other experimenters, such as Duparc and coworkers, have used only 1-hour treatments. Valenski, on the other hand, shows curves extending to 30 hours or more, and with nitrogen contents which have become constant in magnitude, and it may be presumed that his results probably represent truly homogeneous solid phases.

The calculations which will be made are on the assumption that the nitriding reaction is



$$K = \frac{A_{\text{Mn}_3\text{N}_2}}{A_{\text{N}_2} \cdot A_{\text{Mn}}^3}$$

Selecting experiments where the pressure of nitrogen was 1 atmosphere, one may write

$$K = \frac{\text{m.f.}(\text{Mn}_3\text{N}_2)}{\text{m.f.}^3(\text{Mn})}$$

where the activities have been replaced by the corresponding mole fractions.

The significant data, obtained under conditions seeming to insure homogeneity of the solid phase, are recorded in table 60, and plotted on the conventional $\log K$ v. $\frac{1}{F}$ chart in figure 17. The points so plotted fall

upon two straight lines for the solid material, with a single point representing the liquid phase. That the intersection of these lines should be so close to the upper limit of Haag's γ phase, as indicated by the dotted line, must be more than a coincidence. It seems then that the upper line represents amorphous solid solutions, its lower continuation mixed crystals, but that the chemical behavior of each is adequately explained by the single assumption of Mn_3N_2 as the nitride phase.

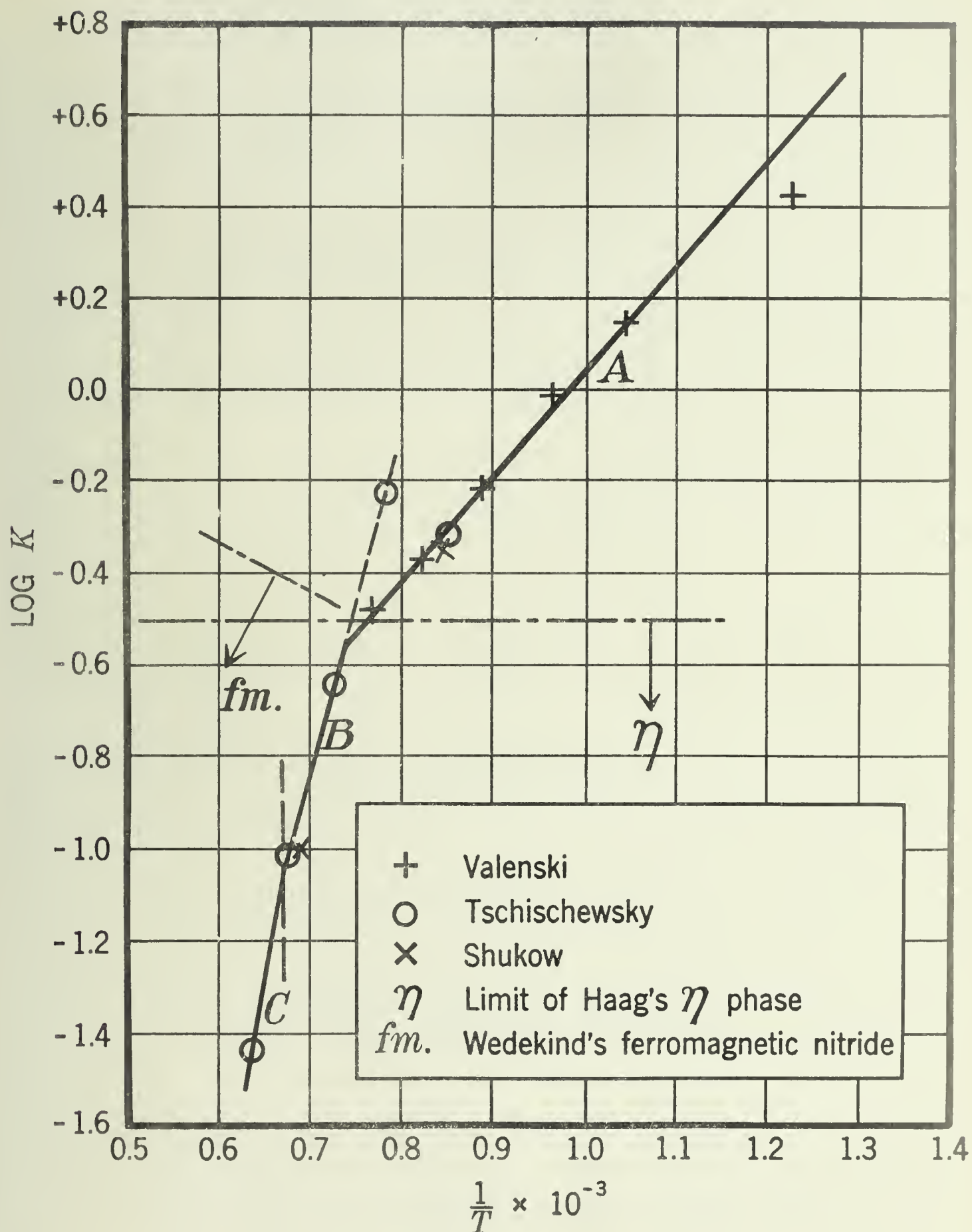


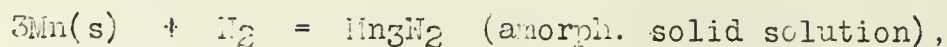
Figure 17.- Equilibrium data for manganese nitrides: A, Amorphous solid solution; B, mixed crystals; C, liquid.

TABLE 60. - Nitriding of Manganese

Experimenter	Temperature		$\frac{1}{T} \cdot 10^3$	N, percent	Mn ₂ N ₂ percent	m. f.		\underline{K}	log \underline{K}
	°C	°K				Mn ₂ N ₃	Mn		
Valenski	540	813	1.270	10.73	74.0	0.448	0.552	2.66	0.425
	620	953	1.049	9.66	66.6	.362	.638	1.394	.144
	760	1033	.968	8.93	61.5	.313	.687	.965	-.016
	850	1123	.891	7.92	54.6	.255	.745	.616	-.211
	940	1213	.825	7.02	48.4	.211	.789	.430	-.366
	1030	1303	.767	6.37	43.9	.182	.818	.332	-.479
Tschischewsky ¹	900	1173	.852	7.3	50.3	.224	.776	.479	-.320
	1000	1273	.785	7.8	53.8	.250	.750	.593	-.227
	1100	1373	.728	5.23	36.1	.139	.861	.218	-.642
	1200	1473	.678	3.29	22.7	.077	.923	.098	-1.012
	1300	1573	.636	1.55	10.7	.033	.967	.037	-1.438
Shukow ²	910	1183	.846	7.16	49.3	.217	.783	.445	-.352
	1175	1448	.691	3.28	22.6	.077	.923	.102	-.992

1 Omitting low temperatures. 2. 1 atmosphere.

From the slopes of the lines and a point on each curve, the following free-energy equations may be deduced:



$$\Delta H = -10,300,$$

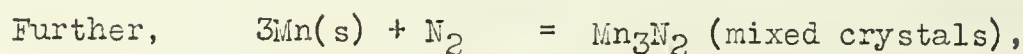
$$\Delta F_{1,213} = -4.578 \cdot 1,213 \cdot (-0.316)$$

$$= +2,030 = -10,300 + 1 \cdot 1,213,$$

$$\underline{I} = +10.16,$$

$$\Delta F_{\underline{T}}^0 = -10,300 + 10.16 \underline{T},$$

$$\Delta F_{298}^0 = -7,270.$$



$$\Delta H = -33,460,$$

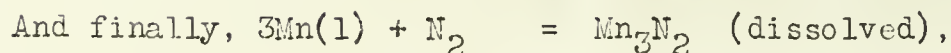
$$\Delta F_{1,373} = -4.578 \cdot 1,373 \cdot (-0.642),$$

$$= +4,035 = -33,460 + \underline{I} \cdot 1,373,$$

$$\underline{I} = -27.32,$$

$$\Delta F_{\underline{T}}^0 = -33,460 + 27.32 \underline{T},$$

$$\Delta F_{298}^0 = -25,315.$$



$$\Delta F_{1,573} = -4.578 \cdot 1,573 \cdot -1.438,$$

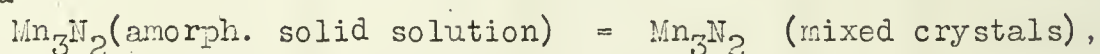
$$= +10,350 = -45,800 + \underline{I} \cdot 1,573,$$

$$\underline{I} = +35.70,$$

$$\Delta F_{\underline{T}}^0 = -45,800 + 35.70 \underline{T},$$

$$\Delta F_{298}^0 = -35,160.$$

From the first and second reactions above the heat of crystallization may be calculated



$$\Delta H = -23,160,$$

$$\Delta F_{\underline{T}}^0 = -23,160 + 17.16 \underline{T},$$

and further from the second and third reactions the heat of fusion is calculated to be $\Delta H = +12,340$. This may be compared with the heat of fusion of 3 gram-atoms of manganese metal = $3 \times 3,576 = 10,728$. The former figure is composite, since the system contains both manganese and its nitride and should presumably be somewhat greater than the heat of fusion of the manganese metal only. The order of magnitude may be taken as confirmatory.

Direct determinations of the heat of formation of manganese nitride have been made by Neumann, Krüger, and Haebeler (272). These authors made direct determinations of the heat evolved by the nitriding of manganese in a bomb and indirectly by the oxidation of an externally prepared nitride sample. They assume Mn_5N_2 to be formed, largely because they obtained about 8 percent nitrogen in the first experiments. The value given is

$$\Delta H = -57,180 \pm 400 \text{ for the reaction } 5Mn + N_2 \rightarrow Mn_5N_2.$$

These results are based upon a difference method, where the calorimetric ΔT for the reaction was approximately 0.05° , for a determined rise of $1.25^\circ C.$, calculated by difference. Under these circumstances, it is difficult to see how any precision could have been obtained. Moreover, the phase changes disclosed by Haag's diagram lead to the supposition that their product was of indefinite composition.

In the oxidation experiments, the authors seem to have made a serious error. Their manganese was prepared by distillation of electrolytically prepared amalgam in hydrogen, and analyzed 99.75 percent manganese. They state the remainder to be nearly all hydrogen, but they do not correct (so far as their data show) for the heat evolved by oxidation of this hydrogen. Thus, they had the heat of oxidation of 1 gram of metal = 1,845.4 calories, and for a corresponding amount of nitride (5.1 percent N) 1725.5 calories. One gram of metal would contain 0.0025 gram H_2 ; 2 grams of hydrogen produce 57,820 calories when burned to water vapor at $25^\circ C.$, then the heat of 0.0025 gram = $\frac{0.0025}{2} \times 57,820 = 72.3$ calories. Then 0.9975 gram of Mn produces $1,845.4 - 72.3 = 1,773.1$ calories, or 1,777.5 per gram. One gram of Mn corresponds to 0.0537 gram of N_2 , whence the heat of nitriding is $\frac{1775.5 - 1725.5}{0.0537} \times 0.28 = 26,100$,

instead of the 62,400 calories per mole calculated by the authors. This figure is at least the same order of magnitude as the 33,460 calories calculated for mixed crystals. Since this sample was prepared separately at $600^\circ - 700^\circ C.$, from the bomb experiments it may be assumed that it was essentially homogeneous, and it falls well within the limits of Haag's η phase.

The ferromagnetic properties of manganese nitride have caused considerable interest. It is worthy of note that Wedekind (380) has given the limits of preparation of the ferromagnetic material as over $1,200^{\circ}\text{C}.$, and corresponding to 6.8 percent N. If this point is placed on the log

K v. $\frac{1}{T}$ chart, it appears that the ferromagnetic field corresponds to the true mixed crystals or the η phase of Haag and furnishes final confirmation of the point of view adopted.

The data of Duparc have not been shown on the chart, but they approach asymptotically to the lines given and are obviously results not obtained at equilibrium because of the short time periods used.

Explanation of the various data presented above has but one serious flaw; it does not include experiments which have been made at various pressures. Instead of K varying inversely as the first power of the nitrogen pressure, the data available (Shukow and Duparc) indicate that K seems to be a function of the square or cube root of the N_2 pressure. Since one cannot be certain that equilibrium in the solid phase had been obtained, it may be supposed that the pressure experiments were not, in fact, correct, and that more precise results might corroborate the assumptions made.

The variation of solubility of gas in metal according to a square- or cube-root equation has recently been discussed by Gottschalk and Dean (121). It should be noted in connection with the manganese nitride that the chemical-compound relationships seem now fairly clear, but that the pressure effect may be explained if only associated molecules react; and not dissociated molecules or atoms, as some authors suppose.

CONCLUSION

For the convenience of the user of the data which have been collected in this chapter, the figures for specific heats, free energies, entropies, and heats of formation used or found have been summarized tables 61 to 64. Before such figures are used for important calculations, reference should be made to the original calculations to indicate the limitations of the available data and in consequence of the derived free-energy expressions.

It should not be forgotten, in this connection, that the calculation of equilibria from free-energy data suffers from a considerable increase in percentage error when the resultant free energy must be obtained by the summation or difference of large individual quantities. Under such circumstances, errors of several hundred percent are possible, and the calculations may not give more than the order of magnitude of the true result. It is perhaps a surprising and reassuring circumstance that the degree of correlation obtained in the calculations is as good as indicated. The calculations seem to have brought the manganese systems to such a state that a few well-performed crucial experiments would authenticate the entire list of properties.

Referring more specifically to the tables, the reader will note that most of the figures in table 61, showing the specific heat values used in this work, are estimated from data near room temperature. Where a third coefficient is given, it must not be forgotten that the quadratic form used, $C_p = a + bT + cT^2$, must necessarily have a maximum or minimum at some temperature specified by the relationship

$$\frac{dC_p}{dT} = 0 = b + 2cT,$$

$$T_{\max} = -\frac{b}{2c}$$

It is practically certain that true specific heat values do not actually show such maximum points, and the quadratic form of expression is only even approximately correct when T_{\max} falls outside of the range of validity of the expression. These maxima occur for the various materials at the following approximate temperatures: MnO , $1,150^\circ C$; Mn_2O_3 , 750° ; Mn_3O_4 , $1,000^\circ$; MnO_2 , 500° ; MnS , $1,100^\circ$; $MnCO_3$, $2,100^\circ$. For the first five materials above the criterion is not satisfied, and the formulas given must not be supposed actually to represent true specific heats. The best that can be said for them is that they represent an attempt to estimate such values in the complete absence of data at high temperatures in a manner that leaves the limiting values consistent with available information.

The real difficulty here is obviously the unsuitability of the quadratic form. Unfortunately, no better substitute in the form of other analytic expressions has so far been devised.⁵ Had one actual data, a preferable method of calculation would be a tabular method similar to that used for the volatilization of manganese.

In the remaining tables of free energies, entropies, etc., some attempt has been made to estimate the apparent uncertainty of the figures. Especially as regards free-energy values, these estimates are based on the writer's judgment and prejudices. It is hoped that the latter have been on the conservative side, but here again it must be pointed out that all that is claimed is that the figures given usually fit the actual available data in the range where the results are supposed to be valid. In the case of the dissociation of oxides, even this last criterion is not fulfilled, other estimates, such as the entropy of Mn_2O_3 , requiring some violence to actual dissociation figures.

⁵ Since this was written the author, with K. K. Kelley, devised a superior form of analytical expression, which appeared in an article entitled "An Equation for the Representation of High-Temperature Heat Control", in the Journal of the American Chemical Society, Vol. 54, 1932, P. 3243.

TABLE 61, - Summary of Specific-Heat Values Used in this Work

$$(\underline{C}_p = \underline{a} + \underline{bT} = \underline{cT^2})$$

Material	Form	<u>a</u>	<u>b</u>	<u>c</u>	Apparent valid temperature range, °C.
Mn	α	3.78	$47.38 \cdot 10^{-5}$		0 - 835
Do	β	9.00			835 - 1,044
Do	γ	10.00			1,044 - 1,220
Do	Liquid	11.30			1,220 - 1,400
MnO ^a	Solid	7.43	$1.038 \cdot 10^{-2}$	$-3.62 \cdot 10^{-6}$	0 - 1,650
Mn ₂ O ₃ ^a	do	10.33	$5.3 \cdot 10^{-2}$	$-2.57 \cdot 10^{-5}$	0 - 1,100
Mn ₃ O ₄ ^a	do	19.25	$5.38 \cdot 10^{-2}$	$-2.09 \cdot 10^{-5}$	0 - 1,500
MnO ₂ ^a	do	1.92	$4.71 \cdot 10^{-2}$	$-2.97 \cdot 10^{-5}$	0 - 500
MnS ^a	do	10.21	$6.56 \cdot 10^{-3}$	$-2.42 \cdot 10^{-6}$	0 - 1,610
Do ^a	Liquid	16.			1,610 -
MnCO ₃ ^a	Solid	7.79	$4.21 \cdot 10^{-2}$	$-.90 \cdot 10^{-5}$	0 - 500
MnSO ₄ ^a	Anhydrous	22.5	$1.5 \cdot 10^{-2}$		0 - 1,000
MnCl ₂ ^a	Solid	16.33	$5.0 \cdot 10^{-3}$		0 - 650
Do ^a	Liquid	24			650 -
Do ^a	Vapor	12	$2 \cdot 10^{-3}$		650 -

^a Estimated from low temperature data, etc.

TABLE 62. - Summary of Standard Free Energies of Formation

Material	Reference state	Temperature, °K.	ΔF°
Mn(β)	Mn(α)	298	+456
Mn(γ)	do	298	$\frac{1}{1}$ -42
Mn(l)	do	298	$\frac{1}{1}$ -3,533
Mn(g)	do	298	+48,913
Mn ⁺⁺	Mn(α)?	291	-52,450 \pm 500
MnO	Mn(α), $1/2$ O ₂	298	-91,310 \pm 700
Mn(OH) ₂	Mn(α), O ₂ , H ₂	291	-145,960 \pm 1,000
Do	MnO, H ₂ O	291	+1,860 \pm 1,000
Mn ₃ O ₄	3Mn(α), 2O ₂	298	-319,820 \pm 500
Mn ₂ O ₃	2Mn(α), $3/2$ O ₂	298	-215,040 \pm 3,000
MnO ₂	Mn(α), O ₂	298	-108,610 \pm 1,000
MnS	Mn(α), S(rh)	298	-47,025 \pm 500
MnCO ₃	MnO, CO ₂	298	-15,913 \pm 300
Do	Mn(α), C(gr.), $\frac{3}{2}$ O ₂	298	-201,000 \pm 1,000
MnSO ₄	Mn(α), S(rh), 2O ₂	298	-226,935 \pm 1,500
MnCl ₂ (g)	MnCl ₂ (l)	298	+31,972
MnCl ₂ (s)	Mn(α), Cl ₂	298	-102,107 \pm 500
MnCl ₂ (l)	MnCl ₂ (s)	298	+3,603 \pm 500
Do	Mn(α), Cl ₂	298	-98,504 \pm 1,000
Mn ₃ C ₄ (in sol.)	3Mn(l), 4C(s)	298	+3,705
Mn ₃ C ₄ (solid sol.)	3Mn(s), 4C	298	-32,810
Mn ₃ N ₂ (amorph. solid sol.)	3Mn(s), N ₂	298	-7,270
Mn ₃ N ₂ (mixed cryst.)	3Mn(s), N ₂	298	-25,315
Mn ₃ N ₂ (dissolved)	3Mn(l), N ₂	298	-35,160

- ¹ These values should obviously be positive, since a negative value would seem to indicate the product stable with respect to alpha manganese at 298°K. The difficulty is due to the long extrapolation of specific heat values, beyond the range where such values are truly valid. These figures are to be regarded simply as formal coefficients in the standard free energy equation, the use of which will make calculated values, in the range where the specific heat values are valid, fit the known experimental facts.

TABLE 63. - Summary of Free-Energy Equations

$$(\Delta F_T^0 = \Delta H_0 + A T \ln T + BT^2 + CT^3 + I T)$$

Product	Reference State or Reactant	ΔH_0	A	B	C	I
Mn(β)	Mn(α)	-1,102	-5.22	+3.69 $\cdot 10^{-3}$	-	+33.52
Mn(γ)	Mn(β)	-1,073	-1.0	-	-	+ 7.99
Mn(l)	Mn(γ)	-5,516	-1.30	-	-	+11.01
Mn(g)	Mn(l)	+65,930	+6.33	-	-	-77.92
Mn(γ')	Mn(α)	-2,175	-6.22	+3.69 $\cdot 10^{-3}$	-	+41.46
Mn(l)	Mn(α)	-7,691	-7.55	+3.69 $\cdot 10^{-3}$	-	+52.47
Mn(g)	Mn(α)	+58,239	-1.22	+3.69 $\cdot 10^{-3}$	-	-25.45
MnO	Mn(α), 1/2 O ₂	-96,700	- .40	-1.25 $\cdot 10^{-3}$	+0.60 $\cdot 10^{-6}$	+20.58
Mn ₂ O ₃	2Mn(α); 3/2 O ₂	-234,370	+6.98	-1.83 $\cdot 10^{-2}$	+ .428 $\cdot 10^{-5}$	+30.15
Mn ₂ O ₄	3Mn(α), 2O ₂	-344,627	+5.09	-1.485 $\cdot 10^{-2}$	+ .332 $\cdot 10^{-5}$	+58.35
MnO ₂	Mn(α), O ₂	-120,230	+8.36	-1.94 $\cdot 10^{-2}$	+ .495 $\cdot 10^{-5}$	-3.31
2Mn ₂ O ₇	4MnO ₂ , O ₂	+12,296	-19.48	+4.22 $\cdot 10^{-2}$	-1.123 $\cdot 10^{-5}$	+72.80
4Mn ₂ O ₄	6Mn ₂ O ₃ , O ₂	+29,027	-21.52	+5.09 $\cdot 10^{-2}$	+1.177 $\cdot 10^{-5}$	+45.67
MnS	Mn(α), S(rh)	-46,313	-2.31	+2.76 $\cdot 10^{-3}$	+ .403 $\cdot 10^{-6}$	+9.90
MnCO ₃	MnO, CO ₂	-27,245	+7.34	-1.32 $\cdot 10^{-2}$	+ .76 $\cdot 10^{-6}$	+ .10
MnCl ₂ (g)	MnCl ₂ (l)	+46,950	+12.0	-2.10 $\cdot 10^{-3}$	-	-118.27
MnCl ₂ (s), H ₂ S	MnS, 2HCl	-27,481	+ .15	- .451 $\cdot 10^{-3}$	- .36 $\cdot 10^{-6}$	+32.64
MnCl ₂ (l)	MnCl ₂ (s)	+ 2,206	-7.62	+2.5 $\cdot 10^{-3}$	-	+47.30
Mn ₃ Cl ₄ (liq. sol.)	3Mn(l), 4C	+ 3,090	-	-	-	+ 2.062
Mn ₃ Cl ₄ (solid sol.)	3Mn(s), 4C	-52,780	-	-	-	+67.00
Mn ₃ N ₂ (amorph. solid s.)	3Mn(s), N ₂	-10,300	-	-	-	+10.16
Mn ₃ N ₂ (mix. cryst.)	3Mn(s), N ₂	-33,460	-	-	-	+27.32
Mn ₃ N ₂ (dissolved)	3Mn(l), N ₂	-45,800	-	-	-	+35.70

TABLE 64. - Summary of Entropy and Heat of Formation Values

Material	S_{298}	ΔH_{298}
Mn (α)	7.3 \pm 0.5	0
MnO	14.4 \pm 0.6	-96,500 \pm 650
Mn(OH) ₂	1/ 23.9 \pm 1.5	-165,000 \pm 2,000
Mn ₂ O ₃	1/ 21.1 \pm 2.0	-235,050 \pm 5,000
Mn ₃ O ₄	35.5 \pm 0.7	-345,000 \pm 1,000
MnO ₂	13.9 \pm 0.4	-121,260 \pm 2,000
MnS	13.7 \pm 0.3	-45,890 \pm 300
MnCO ₃	20.5 \pm 0.4	-219,100 \pm 800
MnSO ₄	1/ 33.1 \pm 3.0	-250,735 \pm 1,500
MnCl ₂ (g)	- -	2/ +43,550 \pm 500
MnCl ₂ (s)	1/ 25.1 \pm 2.0	-112,600 \pm 1,000
MnCl ₂ (l)	- -	-108,344 \pm 1,500

1/ Estimates, not directly determined.

2/ From liquid.

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MANGANESE; ITS OCCURRENCE, MILLING, AND METALLURGY

PART III.

CHAPTER 6. GENERAL METALLURGY OF MANGANESE

CHAPTER 7. HYDROMETALLURGY OF MANGANESE

CHAPTER 8. PYROMETALLURGICAL TREATMENT OF MANGANESE ORES



BY

R. S. DEAN, EDMUND S. LEAVER, AND T. L. JOSEPH

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PART III

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1. The first part of the document discusses the importance of maintaining accurate records of all transactions and activities. It emphasizes the need for transparency and accountability in financial reporting.

2. The second part of the document outlines the various methods and techniques used to collect and analyze data. It includes a detailed description of the experimental procedures and the statistical analysis performed.

3. The third part of the document presents the results of the study, showing the trends and patterns observed in the data. It includes several tables and figures that illustrate the findings.

4. The fourth part of the document discusses the implications of the results and the conclusions drawn from the study. It highlights the significance of the findings and their potential applications in the field.

5. The fifth part of the document provides a summary of the key points and a final conclusion. It reiterates the importance of the study and the need for further research in this area.



CHAPTER 6. GENERAL METALLURGY OF MANGANESE

By R. S. Dean²

The metallurgy of manganese has developed along the lines of iron metallurgy. Enough high-grade ore has been available so that by reduction with carbon it could be turned into an iron-manganese alloy and a slag containing some 9 to 15 percent manganese. More siliceous ores have been smelted in the blast furnace to form iron-silicon-manganese alloys called spiegel.

When lower-grade ores came in for serious consideration, which has only been since 1914, two types of processes received the most consideration -- hydrometallurgical processes to produce a concentrate which could be smelted like the higher-grade ore and processes involving concentration of the manganese from manganiferous iron ores by preferential reduction of the iron (184)³.

Special chapters in this bulletin are devoted to the production of ferromanganese from ore, the hydrometallurgy of manganese, and the Bureau of Mines modification of the preferential reduction process.

Other methods of producing manganese alloys from low-grade or refractory ores have been suggested and may find future application.

Manganese has a very high affinity for sulphur and is particularly adaptable to concentration by matte smelting. According to Betts (26), manganese silicate may be smelted with pyrite to give manganese sulphide and iron silicate. This method would be applicable to concentration of either the silicate or carbonate ores. The sulphide, once obtained, could be roasted to oxide and reduced in the usual way.

Another possibility for the production of manganese from silicate ores involves their reduction by silicon or manganese silicides. Swedish Patent 58290, February 25, 1925, covers the reduction of a slag rich in manganese by silicon-manganese alloy containing more than 10 percent silicon, Joseph H. Brennan (38) describes the concentration of manganese iron ores by partial reduction with silicon. Beckett (3,4,19) describes the reduction of manganese dioxide with a manganese silicide for the production of standard ferromanganese. Beckett gives as a typical alloy, manganese 64.4, silicon 24.6, and as a typical ore, 56.50 manganese as MnO₂. The product of the reaction between these two is 80.11 percent manganese and 1.22 percent silicon.

² Chief engineer, Metallurgical Division, U.S. Bureau of Mines.

³ Throughout parts I to IV, numbers in parentheses refer to items in this bibliography, and page numbers to those in the reference cited.

CHAPTER 7. HYDROMETALLURGY OF MANGANESE

By Edmund S. Leaver⁴

Hydrometallurgical methods for utilization of low-grade deposits of manganese ore present a difficult problem in comparison with the usual leaching processes for the recovery of precious metals or even with the process of leaching a 1-percent copper ore. In the concentration of manganese ore by lixiviation methods the solvent must be cheap or through regeneration become available for reuse at low cost. This is easily understood when we consider that the average milling grade of manganese ore will exceed 15 percent in manganese content and will require a corresponding amount of chemical in the solvent solution to react with it to effect dissolution necessary for separation and removal from the ore residue. Other problems are the recovery of the manganese from the solvent solution and the regeneration of the chemicals used. The feasibility of treating manganese ores by hydrometallurgical processes is, of course, governed by the cost of treatment and the selling price of the manganese product. The high cost of dissolving the manganese mineral and the ease with which the solutions can be purified favor the preparation of a high-grade product. Such products include the manganese compounds that enter into the textile industry and the manufacture of electric batteries, glass, pottery, tile, brick, paint, ink, and fertilizers.

LARGE MANGANESE DEPOSITS IN THE UNITED STATES

The principal large bodies of manganese ores known to be available in the United States and thought to be probable sources of supply for some form of hydrometallurgical treatment are listed in table 65.

More details on the occurrence of each of these manganese deposits, including physical properties and results of concentration experiments, are given in chapter 4.

TYPES OF ORES

From a hydrometallurgical standpoint manganese ores may be divided into several types: (1) Ores in which the manganese mineral is readily soluble in at least one solvent; (2) ores in which the manganese mineral requires preliminary treatment to make it soluble; (3) ores containing manganese in insoluble form; (4) ores which destroy excessive amounts of solvents due to soluble impurities; (5) ores which present physical properties that increase the cost of treatment.

⁴ Metallurgist, U. S. Bureau of Mines.

TABLE 65. - Large Manganese Deposits in the United States

(Possible source for hydrometallurgical treatment)

Name and location	Tonnage	Type of ore	Mineral
Kingman district, Ariz.	50,000,000	Oxides in sandstone and shale.	Psilomelane, quartz, calcite, gypsum, feldspar.
Batesville, Ark.	420,000	Ferruginous clay and shale.	Psilomelane and braunite iron oxides.
Leadville, Colo.	3,000,000	Ferruginous in limestone.	Oxides of manganese and iron.
Cartersville district, Ga.	5,000,000	Clay and quartzite.	Psilomelane.
Cuyuna, Minn.	30,000,000	Finely disseminated ferruginous, siliceous chert.	Psilomelane hematite.
Butte, Mont.	400,000 (1)	Siliceous oxides. Siliceous carbonate.	Pyrolusite. Rhodochrosite.
Philipsburg, Mont.	1,780,000	Quartz gangue.	Pyrolusite and other oxides.
Las Vegas, Nev.	285,000	Clay and tuff.	Wad and pyrolusite.
Little Florida Mountains, N.Mex.	700,000	Felsitic rhyolite.	Wad; pyrolusite and psilomelane.
Silver City, N. Mex.	1,000,000	Dolomite lime.	Pyrolusite, manganite, and hematite.
Chamberlain, S. Dak.	100,000,000	Shale with nodules of ----	Carbonates of manganese iron, lime.
Blue Ridge, Va.	11,000,000	Siliceous clay.	Psilomelane.

1/ Very large.

TABLE 65. - Large Manganese Deposits in the United States--Continued

Name and location	Analysis, percent				Remarks
	Mn	Fe	P	SiO ₂	
Kingman district, Ariz.	8.1	2.3	0.05	68.5	Flotation not satisfactory; gravity concentration poor.
Batesville, Ark.	6.6	7.5	.30	32.3	Flotation no hope; gravity not successful.
Leadville, Colo.	19.6	20.4	.04	7.41	Better grade, shipped to Pueblo.
Cartersville district, Ga.	31.2	16.8	.16	10.2	Concentrating and shipping the better grade ore.
Cuyuna, Minn.	15.0	29.5	.03	25.2	Concentration difficult.
Butte, Mont.	12.3	2.4	.04	75.0	Flotation negative.
Philipsburg, Mont.	21.3	2.64		49.4	Flotation good.
	18.0	2.1	.18	53.4	Concentrated with big loss in tails, used battery for crushing.
Las Vegas, Nev.	24.0	2.0	.05	35.0	Difficult to concentrate.
Little Florida Mountains, N.Mex.	22.6	2.7	.14	39.2	Concentrate and ship.
Silver City, N. Mex.	13.0	39.0	.01	8.0	Does not concentrate favorably.
Chamberlain, S. Dak.	16.0	11.0	.44	13.0	Analysis is for nodules.
Blue Ridge, Va.	35.0	3.5	- -	12.0	Concentrated by jigs and tables.

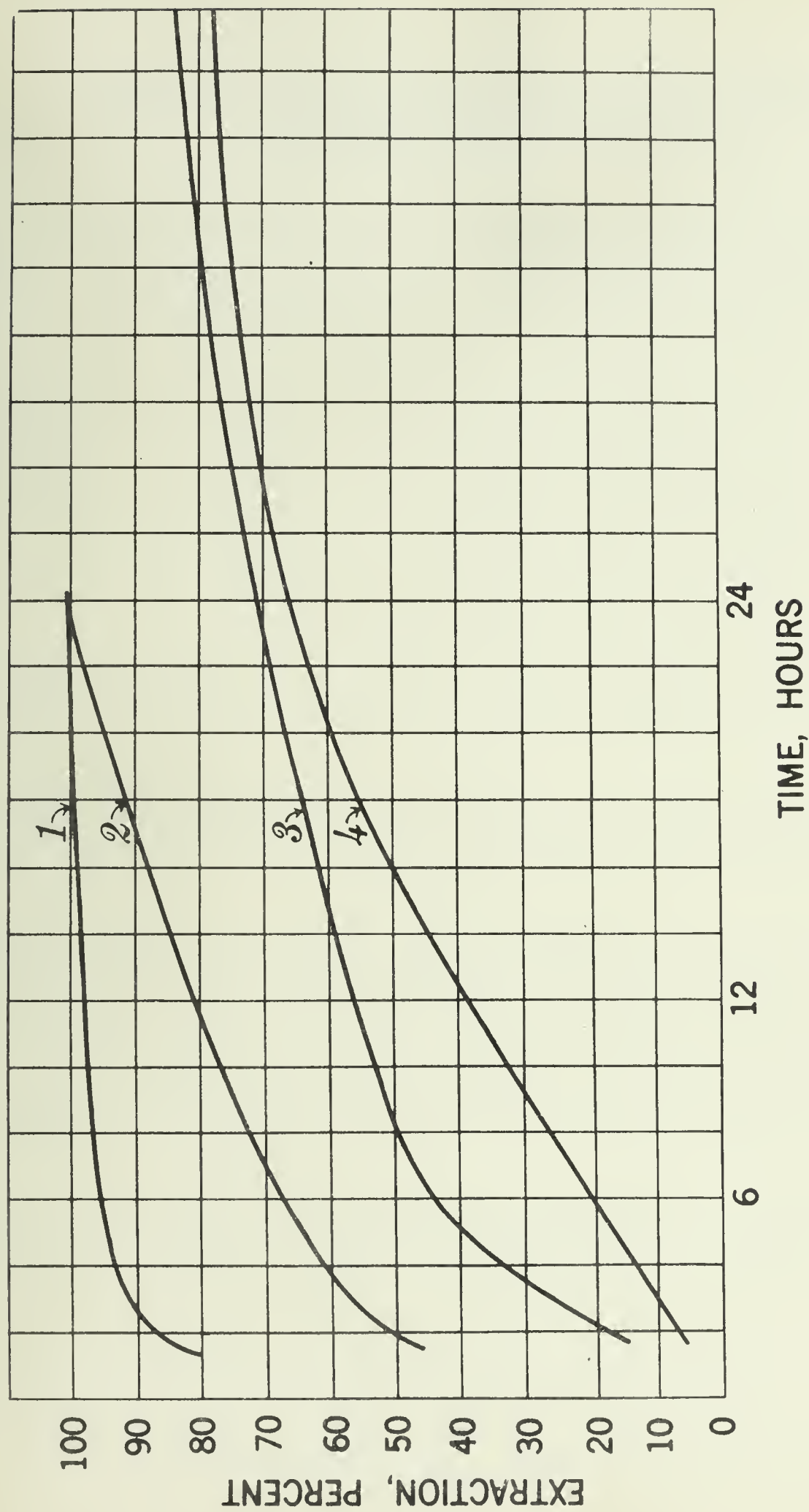


Figure 18.— Rate of dissolution of manganese minerals in 2 percent SO_2 solution: 1, Pyrolusite (–6+10 mesh), psilomelane and others are similar so are not included; 2, hausmannite (–6+10 mesh); 3, manganite (–28+100 mesh); 4, rhodochrosite (–28+100 mesh).

TABLE 66. - Rate of Dissolution of Manganese Minerals
in 2-Percent SO_2 Solutions at Room
Temperature (25°C .)

No.	Sample	Mesh		Treat- ment before leach	Percent extraction of manganese for different time intervals							
		Through	On		20 min- utes	3 hours	6 hours	12 hours	24 hours	2 days	4 days	6 days
1	Pyrolusite ..	6	10	None	21.4	32.5	95.0	97.3	100.0	- -	- -	- -
2	Psilomelane .	6	10	do	26.0	35.5	98.1	100.0	- -	- -	- -	- -
3	Hausmannite .	6	10	do	45.8	57.0	67.4	80.4	100.0	- -	- -	- -
4	Braunite	6	10	do	27.3	32.3	36.0	38.2	100.0	- -	- -	- -
5	Manganite 1/.	6	10	do	7.7	12.8	18.9	28.8	42.0	61.2	84.4	92.5
6	do	28	100	do	14.7	24.3	44.5	- -	70.2	85.1	100.0	- -
7	do	6	10	Roasted	75.5	90.4	96.2	100.0	- -	- -	- -	- -
8	Rhodochrosite	28	100	None	6.0	10.5	20.1	38.5	66.0	80.5	90.5	97.5
9	do	6	10	Roasted	84.8	95.2	98.1	99.2	100.0	- -	- -	- -

1/ The manganite broke into elongated pieces such that although they passed a 6-mesh screen their longest dimension was several times as great as their shortest.

Manganite and rhodochrosite are attacked much more slowly, the smaller particles of manganite (-28+100 mesh) requiring 4 days of dissolution and those of rhodochrosite more than 6 days. Roasting such minerals before leaching might be desirable.

Table 67 contains the results of tests to determine the time required to dissolve manganese minerals and roasted products of different sizes at two different temperatures (for the more refractory minerals). In brief, the results show that decrease in particle size and treatment at 40°C . (instead of at 25°C .) increase the rate of dissolution. They also show that fine grinding, in the case of manganite, and fine grinding and heated solutions, in the case of rhodochrosite, reduce the time of complete dissolution of these minerals to such an extent as to be comparable with the time required to dissolve the manganese from -6+10 mesh samples of the other minerals.

Whether it would be better to use a preliminary roast or to depend on fine grinding and hot solutions for treating the more refractory manganese minerals would have to be determined by large-scale tests of the material under investigation.

TABLE 67. - Effect of Temperature and Size on Rate of Dissolution of Manganese Minerals in 2-Percent SO₂ Solutions

Room Temperature (+25°C.)

No.	Sample	Time for complete extraction					
		-6+10 mesh	-33+100 mesh	-100+200 mesh	-200 mesh	-100+200 mesh	-200 mesh
1	Pyrolusite	16 hours	--	15 minutes	--	--	--
2	Psilomelane ...	12 hours	--	do	--	--	--
3	Hausmannite ...	24 hours	--	2 hours	--	1 hour	--
4	do 1/...	--	--	30 minutes	--	15 minutes	--
5	Braunite	20 hours	--	15 minutes	--	--	--
6	Manganite	2/6 days	4 days	2 days	16 hours	6 hours	4 hours
7	do 1/...	12 hours	--	15 minutes	--	--	--
8	Rhodochrosite .	--	3/6 days	4/2 days	2 days	1 day	11 hours
9	do 1/	18 hours	--	15 minutes	--	--	--
10	Wad	--	--	do	--	--	--

1/ The minerals were roasted at 800°C. to give these products.

2/ 92.5 percent of the manganese extracted.

3/ 97.5 percent of the manganese extracted.

4/ 94.0 percent of the manganese extracted.

DISSOLUTION OF MANGANESE IN ORES IN SULPHUROUS ACID

Tests were made to determine whether dissolution of the manganese minerals of various ores could be effected by sulphurous acid solutions. A partial analysis of the ores, with a statement of their source and the results of treatment with SO₂ solution, is given in table 68.

Samples 1 and 2 were commercial metallurgical ores; samples 3, 4, and 5 were concentrating ores from which chemical grade concentrates were used in the manufacture of batteries; samples 6 and 7 usually are classified as manganiferous iron ores and represent material of which large tonnages have been developed; and samples 8, 9, and 10 are from prospects.

The ores were prepared by grinding to -100-mesh. The conditions of testing were as follows: 25-gram samples, 1 liter of 4 percent (by weight) SO₂ solution, and agitation in stoppered bottles on rolls for 16 hours at 25°C.

The results in table 68 show that a satisfactory amount of manganese was dissolved from nearly all of the ores tested and that in only one instance was the extraction notably poor.

TABLE 68. - Dissolution of Manganese from Ores by Sulphurous Acid

Sample	Source	Percent			
		Mn	Fe	SiO ₂	Mn dissolved
1	Crescent mine, Olympia Peninsula, Wash.	49.6	0.87	13.5	100
2	Livermore district, Calif.	45.2	1.4	20.1	100
3	Trout mine, Philipsburg, Mont.	19.4	1.9	18.3	96.3
4	Climax mine, Philipsburg, Mont.	35.4	1.7	26.2	99.3
5	Moorlight mine, Philipsburg, Mont.	25.4	1.2	35.2	94.7
6	Cuyuna district, Minn.	22.4	33.3	25.2	98.2
7	do	4.1	32.4	37.9	98.1
8	Near Silver City, N. Mex.	16.0	35.6	10.5	97.3
9	Near Midas, Nev.	21.2	2.7	68.1	59.8
10	Near Hopeland, Calif.	19.8	2.5	63.9	98.8
11 ¹ /	Kingman district, Ariz.	6.4	2.3	68.5	100
12 ¹ /	Las Vegas, Nev.	21.6	2.0	55.0	100

1/ Added by author.

It would seem that manganese ores containing the common manganese minerals, except rhodonite, can be successfully treated so that a large part of their manganese content is dissolved in SO₂ solution. The results do not warrant a definite statement as to their rates of dissolution, which, even for the same minerals from different sources, might vary greatly owing to differences in physical or chemical composition or to the presence of impurities that might exert electrolytic effects. This uncertainty is illustrated by the results given previously for different specimens of braunite.

BRADLEY PROCESS (403)

Under the firm name of Bradley-Fitch Syndicate, Ltd., Wilson Bradley and his associates developed in the Lake Superior mining region an adaptation of ammonia leaching that has received much attention. Their results warrant detailed description. The process was intended primarily for leaching siliceous manganese iron ores of the Cuyuna range, Minnesota, but can be adapted for the treatment of the higher oxides of manganese.

An important feature for the recovery of the manganese is a low-temperature (400°C.) roasting of the ore at 65-mesh in a reducing atmosphere formed by the introduction of hydrogen-carbon monoxide gas and water vapor (35, 36). By close regulation of the roast MnO₂ is reduced to MnO and Fe₂O₃ to the magnetic Fe₃O₄. The calcine must be cooled out of the presence of air to prevent re-oxidation. It is then agitated with ammonium sulphate liquor at about 88°C. for 30 minutes. The reduced manganese oxide readily dissolves as manganous sulphate, with generation of ammonia gas. The gas passes to a series of scrubbing towers. The pulp is filtered; and the filtrate containing the manganous sulphate also goes to the scrubbing towers, passing countercurrent

with an excess of ammonia. Under this condition the ammonia reacts with manganous sulphate to regenerate ammonium sulphate and precipitate manganous hydroxide. By the introduction of air with the ammonia gas the manganous hydroxide ($\text{Mn}(\text{OH})_2$) is changed to the more stable manganic hydroxide ($\text{MnO} \cdot \text{OH}$). It may then be settled, washed, dried, and sintered. The ammonium sulphate remaining in solution is reused to leach a fresh charge of the prepared manganese calcine (MnO).

A double system of digestion or leaching is employed; the liquor from the first tank goes to precipitation, and clean solution is used to finish the leach in the second tank, then this solution is used on a new charge in the first tank. By this method iron impurities are precipitated in the first tank by the fresh ore containing excess manganese oxides. The final residue from the second treatment is passed to magnetic separator for the recovery of the iron.

Pilot Plant

The pilot plant installed at the Minnesota School of Mines (fig. 19) had a capacity of about 4 tons per day. It operated with ore of about this composition: Manganese, 17 percent; iron 29.93 percent; silica, 24.42 percent; alumina, 4.14 percent; lime, 1.0 percent; and phosphorus, 0.06 percent.

The ore was crushed with jaw crushers to 4 inches, screened, then crushed with Symons cone to $1/4$ inch and finished with Hardinge ball mill with Dorr classifier in water circuit to pass 65-mesh. The pulp was thickened in a Dorr thickener and dewatered on Dorr filters to about 10 percent moisture. The filter cake was dried in a rotary-type kiln drier using the waste-gas heat from the reducing furnace. A Herreshoff-type furnace was used to reduce MnO_2 to MnO and Fe_2O_3 to Fe_3O_4 , the objective being that the calcine contains the minimum amount of MnO_2 because manganese in this form is not soluble in ammonium sulphate; and the minimum amount of FeO because iron in this form is soluble in ammonium sulphate. The best results were obtained with low-temperature (400°C .) roast. The city gas was used as the reducing agent, and it analyzed:

	<u>Percent</u>		<u>Percent</u>
Illuminants	8.4	H	40.1
CO_2	3.2	CH_4	13.3
O	0.3	N	5.3
CO	28.9		

The selective roasting whereby the reduction of MnO_2 to MnO is nearly complete before the iron reduces to FeO is claimed as a Bradley discovery, and the good control of the roasting operation to prepare the calcine is a necessary condition for good leaching results. The hot calcine is cooled out of the presence of air before going to the leaching system.

Flow Sheet

By following the accompanying flow sheet (fig. 19) details of the process may be better understood. Two iron digesters of a tray type 6 to 18 inches

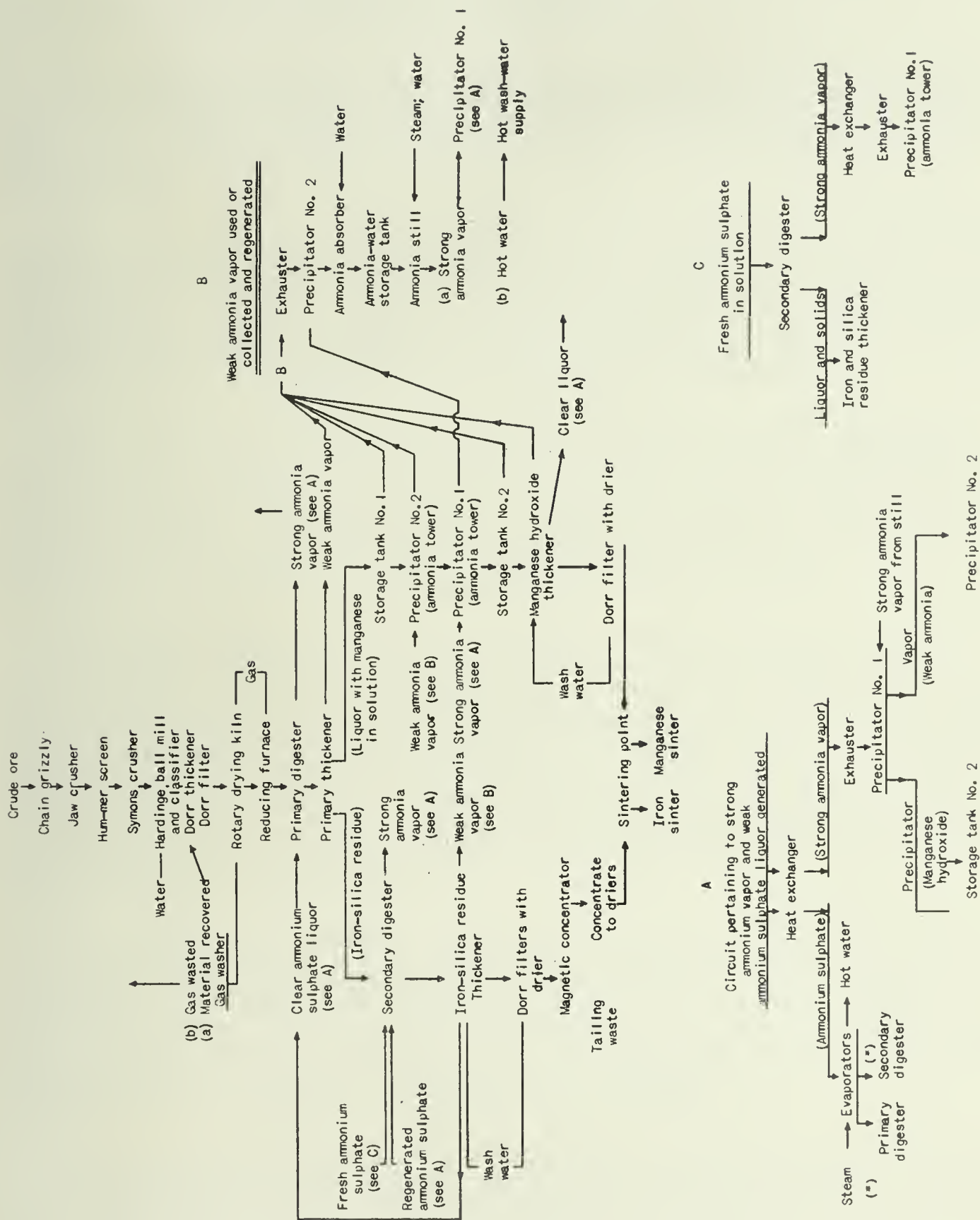


Figure 19.- Bradley process for leaching manganese ores.

in depth, so made that they can be heated on the bottom, are used in the treatment of the ore for dissolution of the manganese. The plan is to allow maximum surface to permit easy expulsion of ammonia vapor.

The prepared calcine goes to the primary digester, where it is agitated with 5 to 15 parts of liquor containing about 10 percent ammonium sulphate, and as this liquor is a wash advanced from the secondary digester it also contains about 4 percent manganese sulphate in solution and small amounts of iron as impurities.

The amount of liquor used in digestion of the ore will vary with the percentage of manganese to be dissolved. For this reason the percentage of ammonium sulphate and manganous sulphate carried in solution is a better guide to the quantity of solution needed for the best results. The pregnant solutions are held near 6 percent manganous sulphate and 7 percent ammonium sulphate. Only 50 percent of the manganese is precipitated in each cycle.

The time of treatment in the primary digester is about 30 minutes at 88°C. to obtain about 60-percent extraction of the manganese. The ammonia vapor expelled during treatment in the digesters is drawn into the ammonia tower to be used for precipitating manganese from the sulphate solution and for regenerating ammonium sulphate. Most of the iron entering as impurities in the liquor advanced from the secondary digester is precipitated by the excess manganese that remains in the pulp during the treatment in the first digester.

The partly treated pulp from the primary digester, containing in solution about 6 percent manganous sulphate and about 7 percent ammonium sulphate, flows to a separator or filter; the clean solution goes to the ammonia tower and there passes countercurrent with ammonia vapor for the precipitation of about half of its manganese as manganous hydroxide; oxygen (in the air) is admitted with the ammonia vapor, or the precipitate is agitated in contact with air to produce the stable manganic hydroxide (MnO.OH), which is removed by filter and sintered to obtain a product as high as 70 percent manganese. The residue from the filtration of the pulp from the primary digester goes to the secondary digester for continued treatment.

In the secondary digester the partly leached ore is further treated, as in the primary digester, except that liquor from the tower after precipitation plus about 1.5 percent fresh ammonium sulphate is used to insure maximum removal of the manganese. This solution contains about 11.5 percent ammonium sulphate plus 3 percent manganese sulphate. After treatment and filtration the solution is advanced for use in the first digester. The leached tails go to magnetic concentrators for recovery of magnetic iron (Fe_3O_4).

During the leaching periods the solutions at 88°C. dissolve manganese and correspondingly lose free ammonia, and both of the reactions are reversed in the precipitator; that is, a large volume of 10 percent ammonium sulphate solution takes up about 2 percent manganese sulphate in addition to the 4 percent it then held, making a 6-percent pregnant solution, and during this additional dissolution correspondingly loses ammonia, reducing the ammonium sulphate

content to about 7 percent. Both go to the cooler towers, where passing countercurrent the ammonia precipitates manganese and reenters the solution as ammonium sulphate. The exit solution from the towers containing about 10 percent ammonium sulphate and 3 percent manganese sulphate is built up by the addition of 1.5 percent fresh ammonium sulphate and then used in the second digester to take up virtually all of the soluble manganese remaining in the ore, and in doing so the solution is built up by an additional 1 percent of manganese sulphate.

Washing

Both the residue from the secondary digester and the precipitates from the towers require washing to recover leach solution containing ammonium sulphate and manganese sulphate. The excess water thus added to the system is an important consideration, as it requires expensive evaporation to maintain solution concentration at the economical point. In washing this residue the best results were obtained by the use of vacuum filters; centrifuges are proposed as likely of development.

Magnetic Concentration

The residues from the treatment of Cuyuna manganese ores were treated over a magnetic log washer with indicated recovery of 70 percent total iron in a product containing 2-1/2 percent manganese.

Sintering

It is proposed to sinter both the manganese and the iron byproducts if they are to be used for metallurgical purposes.

Evaporation

Multiple effect evaporators were favored, but on account of cost the simple type of direct evaporation was adopted.

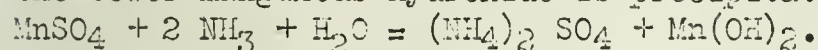
Impurities

In this method of leaching manganese ores the solvent shows a preference for manganese over iron, therefore little or no iron should be dissolved in the primary digester. The iron dissolved during treatment of the low-grade manganese pulp in the secondary digester should, after the solution advances to the primary digester, be mostly precipitated by the excess manganese as ferrous hydroxide and eliminated from solution. If important enough the small amount of soluble iron remaining in the solution may be precipitated with a little ammonia before the solution is advanced to the towers for precipitation of the manganese. Manganese precedes iron in leaching, wherefore iron precipitates more rapidly than manganese from the leach solution.

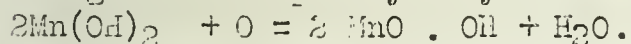
During the leaching a chemical reaction changes the oxide to a sulphate of manganese with the evolution of ammonia gas.



The manganese in solution as manganous sulphate is separated from the residual ore and pumped to a standard ammonia tower, where it passes counter-current with ammonia gas released in the reaction. Under the conditions in the tower manganous hydroxide is precipitated.



The manganous hydroxide reacts with oxygen from the air and readily changes to the partly dehydrated manganic hydroxide.



This stable product is washed, dried, and sintered.

A flow sheet (fig. 20) indicating the principal reactions is given as an aid in following the procedure of the process.

SWEET PROCESSES

Prof. A. T. Sweet and his associates at the Michigan School of Mines, Houghton, Mich., developed two methods for sulphatizing manganese and extracting it from ores. In one method carbonate or lime ores are roasted with ammonium sulphate, and in the other low-lime and relatively high-iron manganese ores are roasted with sulphuric acid to produce soluble manganese sulphates. Both processes depend upon the reaction obtained by roasting the sulphate at a temperature higher than the decomposition point of iron sulphate.

Sweet Ammonia Process

The ammoniacal leach method was developed for beneficiation of the complex carbonate ores that occur along the Missouri river near Chamberlain, S. Dak. The manganese occurs as nodules in shale in the ratio of 1 ton of ore to 10 tons of waste. The nodules or ore show by analysis: Manganese, 16 percent; iron, 11 percent; phosphorus, 0.429 percent; silica, 13 percent; magnesium, 1.87 percent; lime, 15 percent; aluminum, 3.75 percent; loss on ignition, 26 percent. Most of the elements in the ore are intermixed as complex carbonates.

An experimental plant with capacity of 1 ton per day was operated as indicated under the following flow sheet (fig. 21).

A description of the process by Professor Sweet states:

1. The ore was crushed dry. According to the results of a series of tests, crushing to minus 60 mesh gives satisfactory results.
2. The finely crushed material is mixed with the ammonium sulphate, and a little moisture is desirable. The moisture seems to aid in intimately mixing the ore and the sulphate.
3. The mixed pulp is run to the roaster, where the CO_2 is driven off, the sulphate broken up, and the ammonia (NH_3) is liberated. The ammonia and CO_2 are discharged to a tower, collected in the water, and used later on in the process as a precipitant. The sulphate pulp is sent to a countercurrent leaching system.

4. The leach solution from the first thickener of the countercurrent decantation system is clarified and sent to the precipitation tank. The ammonium carbonate solution from the tower is admitted, and the manganese is precipitated as the carbonate, the ammonium sulphate being thereby regenerated.

5. The precipitated manganese carbonate is sent to filtration and sintered.

6. The washed pulp from the final thickener of the countercurrent decantation system is also washed in carbonate solution, steamed, and washed, with the recovery of the ammonium sulphate for the next leach.

The quantity of reagent used in this process is a function of the manganese content and the amount of CO_2 present in the ore. The time of roast may vary from 30 minutes to 45 minutes, depending on the temperature.

This method is particularly suited for the treatment of manganese carbonate ores because the ammonia, on delivering the sulphate ion to the manganese, joins with the liberated CO_2 to form a desirable precipitant for the manganese which was taken in solution. The temperature of the roast should be kept at about 430°C ., as higher temperatures will break up and lose part of the ammonia.

Sweet Acid Process

The second Sweet process is more suitable for ores in which lime and its associated minerals are very low and may contain relatively high iron. The procedure in the operation is easily followed by referring to figure 22.

The steps given by Professor Sweet follow:

(1) The ore was crushed and ball-milled through 65-mesh. The crushing was done dry, though 10 percent moisture in some of the samples did not interfere with the reactions nor with the subsequent roasting operation.

(2) The ground pulp was mixed with sufficient sulphuric acid to take care of the lime and magnesite, together with the small amount of other carbonates which were present. This operation was facilitated by the use of a small concrete mixer. The pulp when properly mixed with the acid was very dry.

(3) The acidified pulp, which was composed of the sulphates of lime, magnesia, manganese, and iron, was introduced into the Herreshoff roasting furnace. This furnace was fired with an oil burner in one of the bottom holes. The temperature at the point of maximum heat was about 790°C .

(4) The roasted product of the furnace, a bright Indian red in color, was cooled and dropped into an agitator, and water sufficient to take

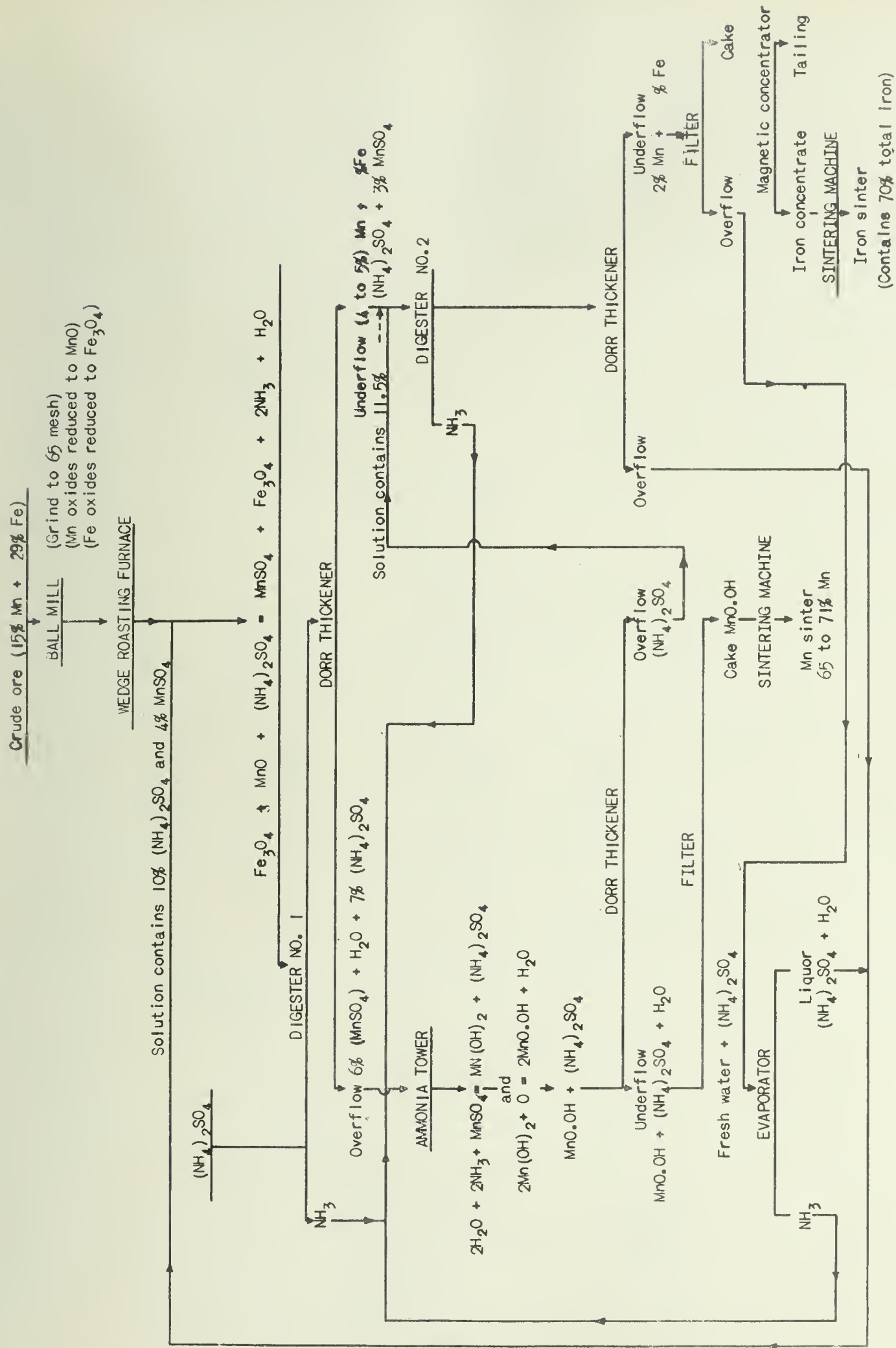


Figure 20.- Bradley leaching process for manganese ores showing principal reactions.



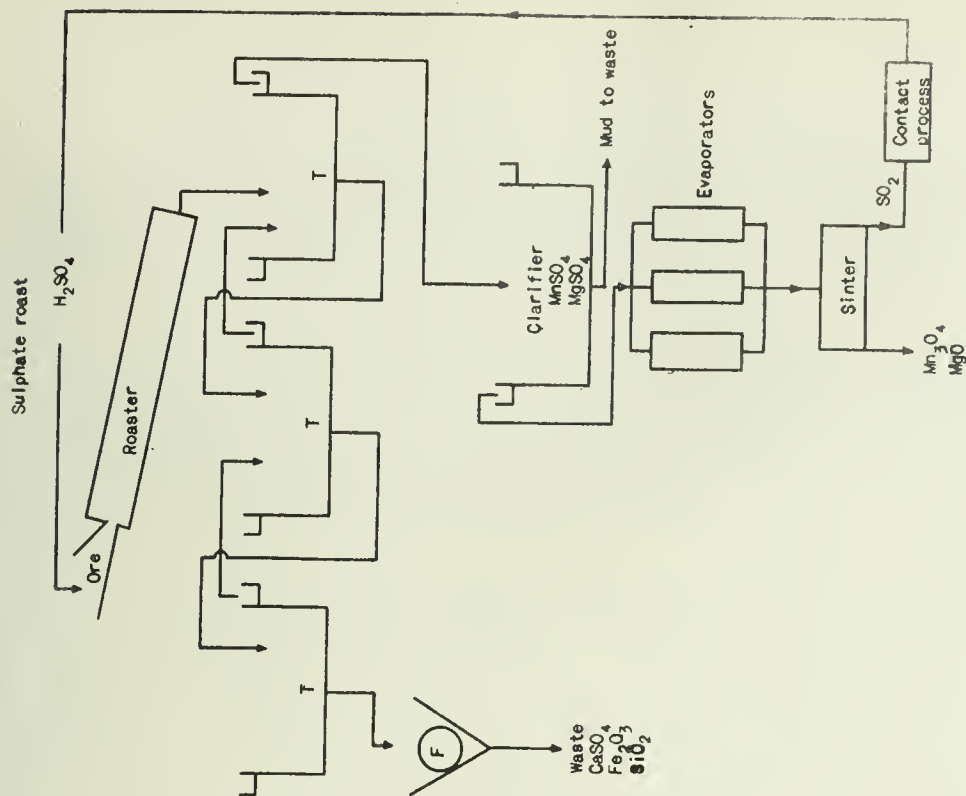


Figure 22.- Sweet acid process No. 2.

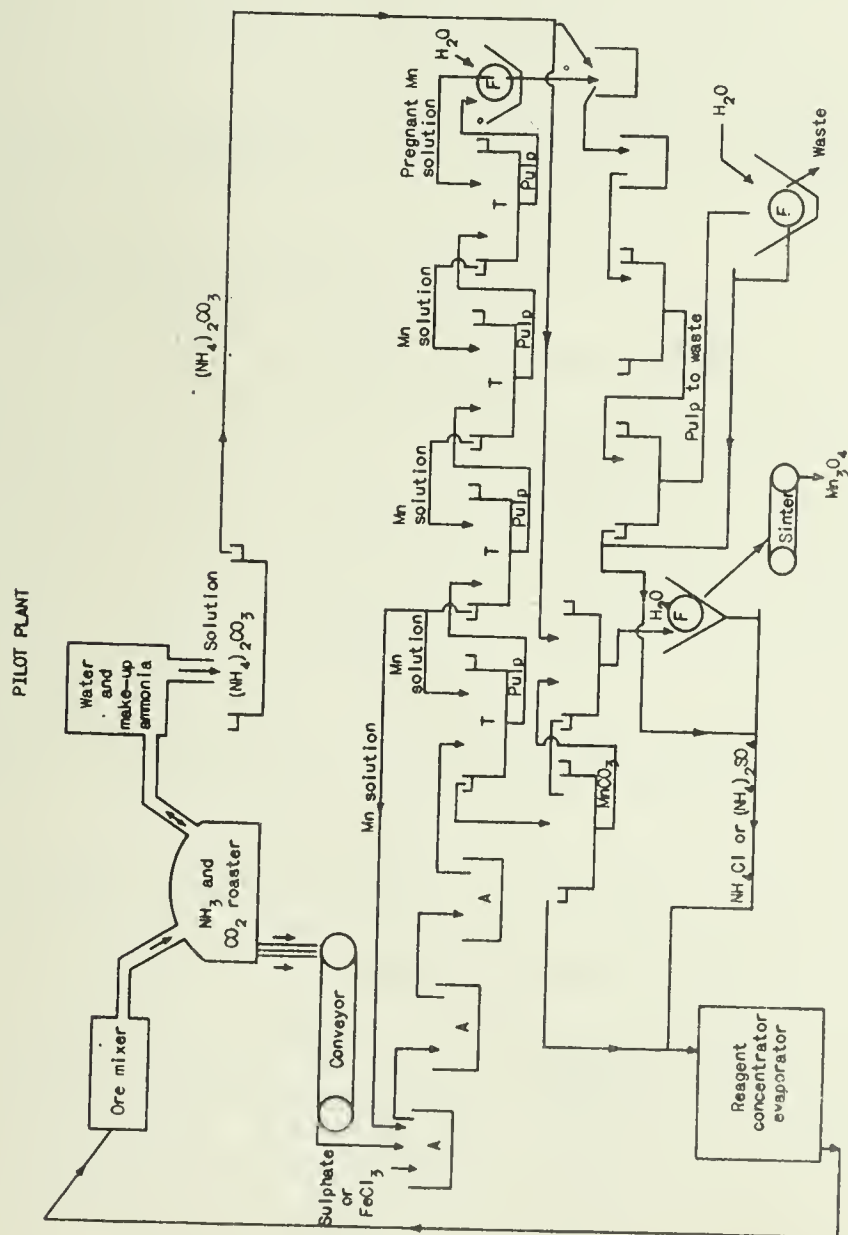


Figure 21.- Sweet ammonia process No. 1 using NH_4Cl or $(NH_4)_2SO_4$.



the soluble salts into solution was admitted. The ore was agitated for about an hour and then decanted to the thickeners, where it was washed by countercurrent decantation. The pregnant solution from the first thickener was discharged to precipitation tanks.

(5) A steam coil was arranged in the precipitation tanks, and the solution was evaporated to precipitate the manganese sulphate. The crystals first forming on the precipitation tank were skimmed off by hand and were found very high in magnosium sulphate. Skimming proved to be one method of removing some of the magnesia.

(6) The manganese sulphate from the precipitation tanks, or more properly from the evaporation tanks, was sent to a Dwight-Lloyd sintering machine, which produced a hard, porous sinter assaying on the average about 60 percent manganese, which was present in Mn_3O_4 . The sinter contained magnesia and some silica. Very little iron was present in the finished product, and the phosphorus was very low. It was impossible to save the SO_3 in the sintering operation in the laboratory, but its reclamation offers only a mechanical difficulty which should be overcome.

The special feature in the process is the roasting with sulphuric acid above the temperature needed to break up any iron sulphate that may be formed. The disadvantages are the cost of acid and the cost of evaporating the sulphate solution.

Typical Examples of Sulphatizing Process

(1) South Dakota Manganosiderites

Sample	Percent				
	Mn	Iron	Silica	Lime	Phosphorus
Head assay	26.0	13.5	10.5	20.0	0.40
Residue	.4	24.0	13.1	50.0	.58

NOTE.- The "residue" in the above and following tables refers to the pulp remaining after the leaching of the manganese sulphate.

This table shows an extraction of well above 90 percent, but because of the high lime content the acid loss is excessive.

(2) Lake Superior Iron Ore

Sample	Percent				
	Mn	Iron	Silica	Lime	Phosphorus
Head assay	17.5	35.2	12.5	.45	0.05
Residue	.3	51.0	15.8	.72	.06

This, of course, is a good extraction, and the low loss of acid due to lime and magnesia is a desirable feature of the ore. The iron contained some sulphur because of the incomplete extraction of the manganese sulphate. Sintering of the finely divided iron, however, is possible and results in the removal of the sulphur.

Sweet Chloride-Leach Process

Professor Sweet also experimented with a third process, which he calls "chloride leach." It is not thought to have any particular advantages over his sulphuric acid process and he does not indicate its importance. The method, with flow sheet (fig. 23), is given because it exemplifies the large amount of thought that has been given to chloride leaching.

The process is best explained in Professor Sweet's words as follows:

Hydrochloric Acid and Soluble Chloride Processes

Numerous attempts have been made to use hydrochloric acid as a solvent for manganese, the chief obstacle in the way being the presence of the calcium compounds in the ore. As in the case of sulphuric acid, the calcium forms a very stable compound with the acid and consumes so much of the reagent that it makes its use prohibitive. With the ore low in lime and allied elements the process has the advantage of being able to extract the manganese without the iron in a very easily controlled operation. As is true when sulphuric acid is used, the addition of raw ore to the pregnant solution from the final thickener reduces the iron content of the solution to practically nothing.

The ore need not be crushed to as fine a mesh as is necessary in the use of sulphuric, for the chloride of lime is very soluble and thus forms no protective coating which delays solution.

Crushing to 10-mesh and subjecting the ore to the leach solution, agitating, thickening, and washing are the steps in the process of removal. Ore is added to the circuit at the point where the pregnant solution leaves for precipitation. The waste which is taken out of the final washing tank may contain enough iron to be a valuable byproduct. Ferric chloride may be substituted for hydrochloric acid with very satisfactory results. Ammonium chloride under special conditions also takes the manganese into solution.

LEAVER DRUM-LEACHING PROCESS

This process is described in Bureau of Mines Bulletin 173. A patent (U. S. Patent 243015) was granted to E. S. Leaver November 18, 1918. It is based on the rapid dissolution of the various higher manganese oxides, especially pyrolusite, psilomelane, and wad, in hot sulphurous acid formed by introducing hot furnace gas containing 2 to 6 percent SO_2 (by volume) counter-current to a pulp flow having a consistency of 2 of water to 1 of ore crushed to about 20-mesh. The carbonate of manganese is made soluble in sulphurous acid by roasting; the silicate is insoluble.

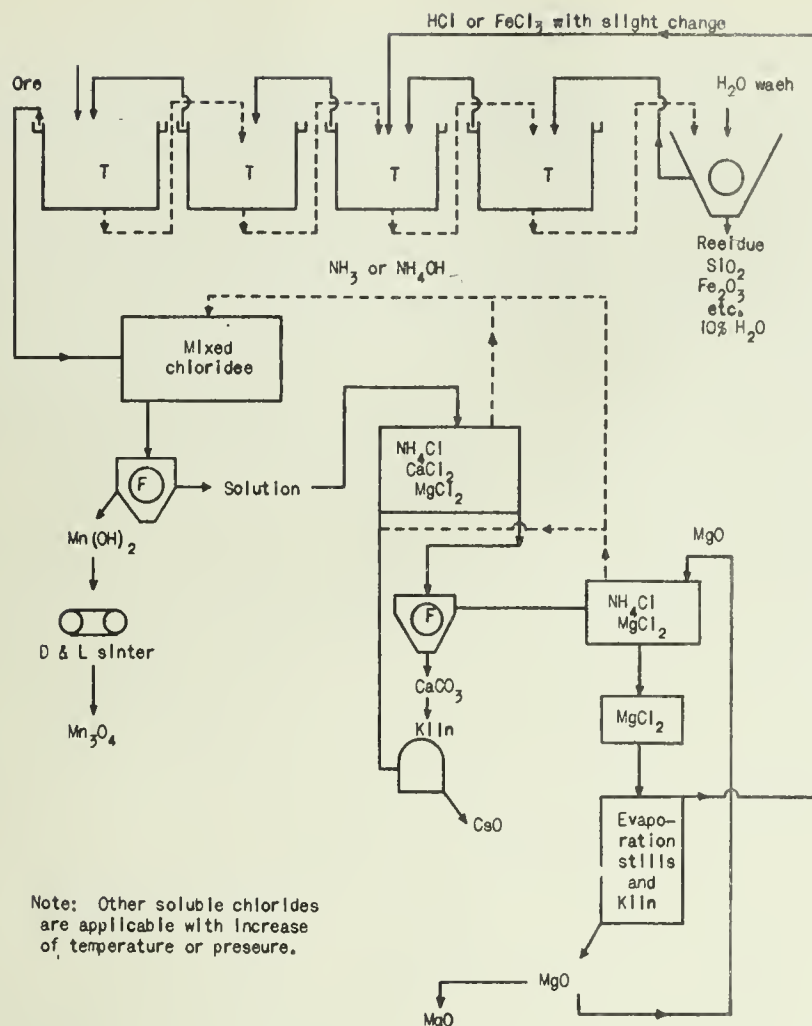


Figure 23.- Sweet chloride process No. 3; chloride leach.

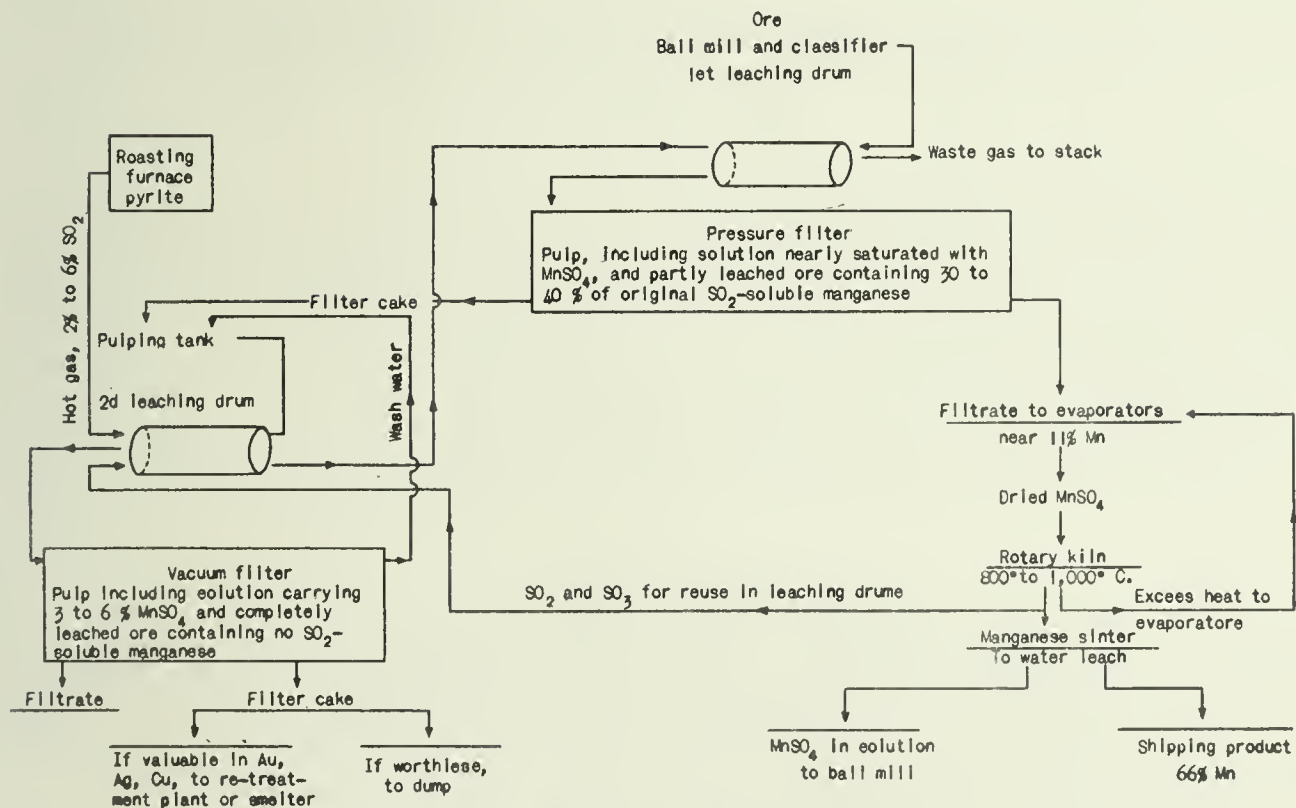


Figure 24.- Leaver drum-leaching process.

The dissolution of the manganese minerals is effected in the Leaver drum (fig. 24). A double-drum system is recommended. The pulp enters one end of the drum, and the hot waste furnace gas containing SO_2 enters the opposite end of the drum. Intimate mixing during the countercurrent flow results in the pulp becoming progressively warmer until it exits at about 45°C ., at which temperature it retains practically no free SO_2 gas. All the dissolved manganese is in the form of the sulphate when the pulp flows from the drum.

By the use of tandem drums the iron and phosphorus that dissolved in removing virtually all the manganese during treatment in the second drum will be precipitated on returning this solution to the new charge in the ball mill or the first drum. This is explained by the changed condition in the operation. The advance solution is used to pulp fresh manganese ore and immediately loses its slight acidity in dissolving manganese. In this neutral solution, the remaining large excess of undissolved manganese, having a greater affinity for the sulphate iron, readily replaces the iron and phosphorus. The precipitated iron and phosphorus become part of the residual ore flowing through the circuit.

The pulp discharge from the first drum should pass to a pressure-type filter for separation of the almost saturated solution (manganese sulphate) and the partly leached ore to be repulped for continued treatment in the second drum. The filtrate is crystallized or evaporated by using waste heat from the furnace. The resultant sulphate is roasted by a direct flame in a rotary-type clinker furnace at a temperature of 825° to $1,000^\circ \text{C}$. The calcined product should run from 60 to 64 percent manganese. The SO_2 and the SO_3 existing with the furnace gas may be reused in the leaching drum. This process, the essential features of which were worked out for the Phelps Dodge Corporation by G. D. van Arsdale and C. G. Maier (U. S. Patent 1348068) was used at the Copper Queen Smelter at Douglas, Ariz., in 1918 on a 10-ton experiment plant for leaching an oxidized manganese-silver ore.

Better understanding of the details in the operation of the proposed double-drum system can be obtained from the flow sheet (fig. 25).

Ball Mill

The oxidized manganese ores or the calcine from the roasted carbonates after primary crushing are fed to a ball mill for wet crushing to 20 mesh in circuit with a classifier. The leach filtrate from the second drum in the circuit is the solution used in the ball mill. It carries 3 to 6 percent of manganese as manganous sulphate and the impurities that are dissolved in completing the leaching of the manganese ore, and it is practically neutral. On contact with a large excess of fresh manganese dioxide ore most of the impurities, including the dissolved iron and phosphorus, are precipitated in the pulp within the ball mill.

Leaching Drum No. 1

The overflow pulp from the ball-mill classifier goes to the first leaching drum for the preparation of an almost saturated solution of manganous sulphate (nearly 11 percent). This is accomplished by intimate contact between the mineral particles and the SO_2 of the furnace gas by breaking the pulp into a fine spray throughout the length of the drum and passing the gas countercurrent with the flow. For better understanding of the operation a description of the drum erected by the Miami Copper Co. is given (fig. 24).

This drum was 7'4" inside diameter by 40'0" inside length and had a rated capacity of 100 tons per day for leaching oxidized copper ores. It was built of fifty-four 3-by 6-inch redwood staves bound with iron hoops and set horizontally, revolving at 14 r.p.m. On the inside center of each stave a groove was cut at an angle of 30° to hold the lifters, which were made of $1\frac{1}{2}$ - by $4\frac{3}{4}$ -inch lumber and fastened to the stave with wooden dowel pins. On the inside length of the drum thirty-eight 2-inch circular vertical baffles were set 1 foot apart and extended to the inside circumference. Each of these baffles contained ninety-nine 4-inch holes set staggered in four circles, each hole in the outer circle set directly under one of the 54 lifters.

The half of each 4-inch hole toward the center was cut square, and through each of the 99 holes in the vertical baffles extended a longitudinal baffle of 2- by 4- by 38-inch lumber, held rigidly in place by wooden pins.

In the center of each vertical disk or baffle a circular hole 24 inches in diameter was provided as a manway for inspection or repair. On the circumference of this 24-inch circle were equally spaced 14 pieces of 2- by 4- by 38-inch lumber, fastened by wooden pins to each vertical baffle. This passageway along the center of the drum was closed at regular intervals by solid 20-inch wooden disks to divert the gas flow and cause part of the gas to flow alternately from the center toward the circumference and back through the next section. The 20-inch disks were easily removed for inspection, although there was little need for entering the drum. The pulp enters the drum through a gas stack provided to exit the waste gas. The movable joint connecting the stack and drum was sealed quite satisfactorily by the use of two 36-inch by $4\frac{1}{2}$ -inch auto tires on which were bolted two machined plates. These plates formed the seal, and the tires allowed for both horizontal and vertical movement. On the pulp exit end, which is also the hot-gas intake, a trap is provided to seal the pulp discharge, and a special design cast-iron seal with asbestos packing joints, allowing for both vertical and horizontal movement was provided by Allis Chalmers Co. The drum was supported by two 18-inch by 8-foot inside-diameter iron castings and held in place by double tapering wedges driven between the outside of the drum and the inside of the casting. The 2-inch vertical disks within the drum were doubled at the point of the carrying castings. On the circumference of each casting 90-pound iron rails were trued to the circle and fastened by set-screws; the circular rails revolved on 30-inch car wheels, two sets on each side in trunnion form.

The drive shaft extended through both sets of car wheels on one side of the drum and by friction between the revolving car wheels and the circular rail satisfactorily turned the drum. The accompanying diagram (fig. 24) shows construction details.

The pulp is lifted by inside buckets similar to a tailings wheel; as the buckets empty and the pulp splashes over each of the longitudinal baffles the SO_2 gas passes directly into the spray through the half-circle opening in the vertical baffles. Under this condition highest absorption of the gas is obtained as the pulp enters one end of the drum cold, this permitting almost complete utilization of the SO_2 before the furnace gas exits at pulp intake of no. 1 drum.

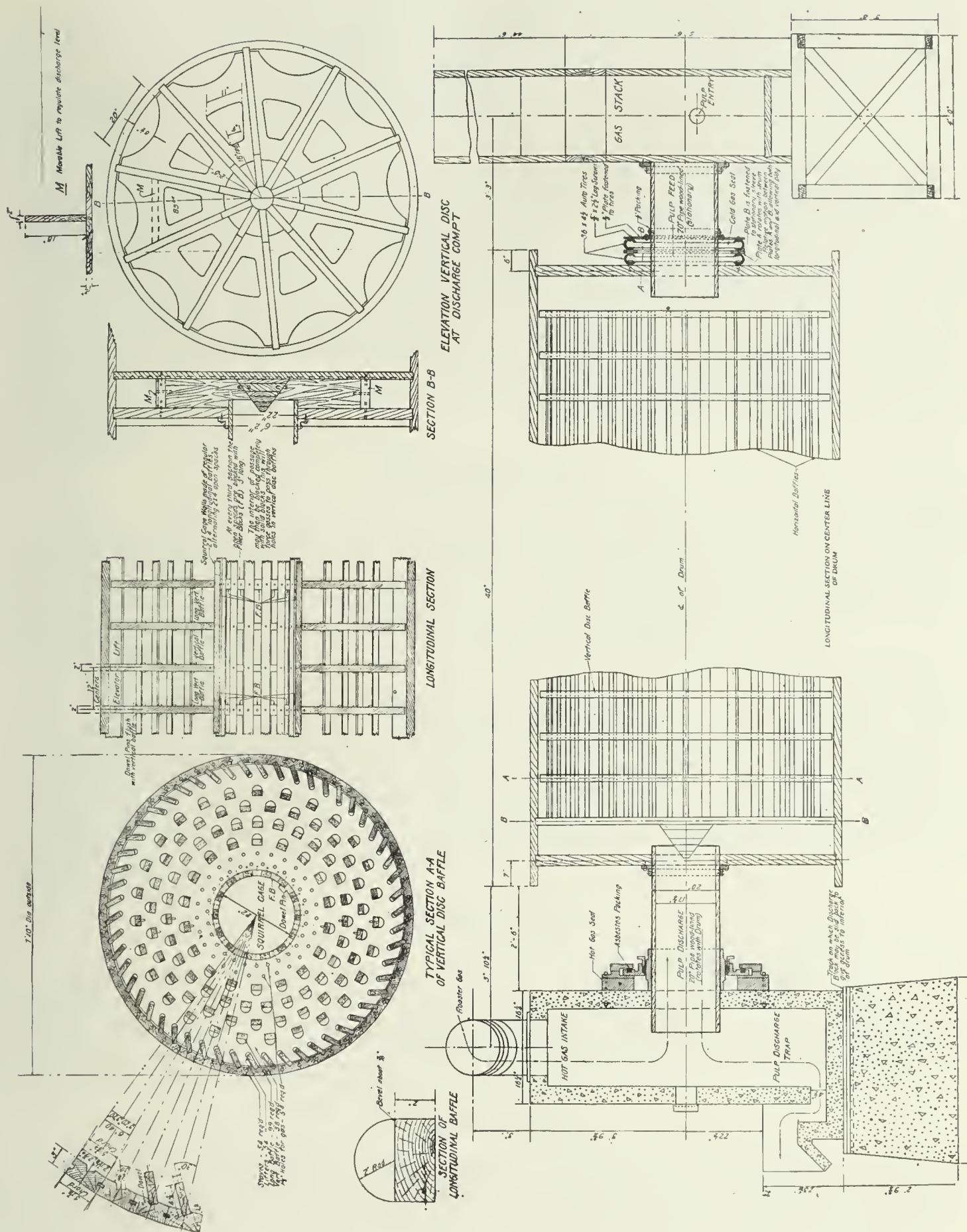


Figure 25.- Details of Leaver SO₂ leaching drum.



During treatment in no. 1 drum it is planned to dissolve only 60 to 70 percent of the soluble manganese. With an excess of readily soluble manganese and SO_2 gas and a limited ratio of solution to ore the pulp exit solution is saturated with manganous sulphate. It is difficult to filter this nearly saturated solution from the pulp; vacuum filters are not satisfactory, and pressure-type filters should be used.

The filter cake from no. 1 drum is repulped with fresh or wash water solution and similarly re-treated in no. 2 drum, except that the hot SO_2 gas direct from the furnace first passes through this drum, and the exit cool gas, still containing a large excess of SO_2 , is used for dissolution of the new charge in no. 1 drum. The pulp exiting from no. 2 drum is hot, about 45°C ., at which temperature the solution contains practically no free SO_2 and is neutral. The solution in this pulp contains only 3 to 6 percent manganese as manganous sulphate and does not present a difficult filtering problem; vacuum filters may be used. The residue is wasted or retained for its gold or silver value. The filtrate goes to the ball mill for reuse on a new charge of ore.

Pregnant Solution

The manganese is recovered from the almost saturated solution of manganous sulphate by crystallization or evaporation. The resulting sulphate is roasted by direct flame in a rotary-type clinker furnace at 825° to $1,000^\circ \text{C}$. The calcine from the roaster is hard, compact clinker. It may contain a small percentage of manganous sulphate, which can be removed by leaching with water. The product at the Douglas plant ran 60 to 64 percent manganese.

In the evaporation of the pregnant solution a problem in corrosion was found to be due to the action of ferrous sulphate and liberated sulphuric acid; leaching in two stages presents a simple solution of this difficulty. As sulphur dioxide has a marked affinity for manganese over that shown for uniting with iron, and as only 60 percent of the manganese in the ore pulp is dissolved in the first drum treatment the remaining 40 percent of manganese insures the dissolution of no iron, and the resultant pregnant liquor will not corrode the iron evaporation pans.

Sulphur Dioxide

The success of this process depends on cheap sulphur dioxide. It may be obtained as a waste product in various smelting operations, also by the roasting of iron pyrite or other sulphides which contain enough lead, copper, silver, gold, or other metals to give the calcine a market value, and by the re-use of the SO_2 and SO_3 in the waste gas from calcining the manganese sulphate.

Plant Control

Analytical methods for plant control, including the determination of SO_2 in pure SO_2 solution, SO_2 and sulphites in drum solution, sulphuric acid content of drum solution, total acidity, SO_2 in furnace gas and acid requirements of ores, are given on pages 77 to 81 of Bureau of Mines Technical Paper 312.

Application

It is recognized that evaporation of the manganese sulphate solution is the expensive part of the proposed process, and conditions must be favorable to compete with imports of high-grade ore. The best chance for success appears as an auxiliary process where the residues may contain other valuable minerals, such as copper, silver, lead, or gold. As carried out at Douglas the process was very nearly a technical and commercial success and might be applicable under favorable conditions in normal times.

WESTLING NITRATE PROCESS

The Westling nitrate process, covered by U. S. Patent 1520079 of July 22, 1924, was used by the American Manganese Products Co. at its plant near Redwood City, Calif. After fine grinding oxidized manganese ores were agitated in tanks similar to the Pachuca type, while the pulp was digested with sulphur dioxide and some air to produce sulphates. The charge was transferred to settlement tanks to remove the coarse particles. The solution and suspended material were separated by an Oliver filter. Powdered calcium carbonate was used to precipitate iron and other impurities from the pregnant solution at 50° C. Calcium nitrate was then added and the solution heated in open pans to separate out the calcium sulphate, which was removed by the Oliver filter. The solution containing manganese nitrate was heated under pressure and thus modified into manganese dioxide and nitrous gases. The nitrous gases were reclaimed by means of scrubbing towers. The manganese dioxide product is of chemical grade, but it did not find a ready market and apparently is not suited for the dry-battery industry.

PROPOSED PROCESSES

The Rare and Precious Metals Experiment Station of the Bureau of Mines has recently conducted small laboratory experiments on combinations of various methods that may aid in improving former proposed processes. Four methods were considered as offering the best possibilities.

No. 1. The original Leaver drum process as given in Bureau of Mines Bulletin 173.

No. 2. Dissolve with waste smelter gas (SO_2) to obtain manganese as sulphate. Precipitate with ammonia plus air to obtain $\text{Mn}(\text{OH})_3$ and ammonium sulphate, filter out manganese hydroxide, boil the filtrate with lime (CaO) to expel ammonia, which is reclaimed by scrubbing towers for reuse, discard calcium sulphate, and calcine $\text{Mn}(\text{OH})_3$ to obtain Mn_xO_y .

No. 3. Dissolve with SO_2 to obtain manganese sulphate. Add calcium chloride while solution is on the ore to make MnCl_2 and CaSO_4 , filter, add $\text{Ca}(\text{OH})_2$ to the filtrate, and agitate with air to precipitate oxides of manganese and regenerate calcium chloride for reuse; calcine the manganese precipitate.

Table 69.- Comparison of four methods suggested for study in improving hydrometallurgy of manganese

Process	(1) Leaver drum SO ₂ leaching	(2) Ammonia	(3) Chloride	(4) Sulphite
Reduction	In first three schemes the original ore is treated direct for dissolution of all minerals except rhodochrosite, which must be given a preliminary oxidizing roast. Rhodonite is untreatable by SO ₂ methods.			Reduce Mn compounds to MnO by reducing roast.
Dissolving manganese	Dissolving is done by SO ₂ plus water in all four schemes. In Rocky Mountain States waste SO ₂ is available. In first three schemes give double leach.		Add CaCl ₂ while in solution on ore to make MnCl ₂ and CaSO ₄ .	Mn in solution as MnSO ₃ ; must exclude air.
Solutions: Purification, Filtration Concentration Transformation	In all cases iron may be precipitated preferentially by ground limestone. MnSO ₄ soluble up to 420 grams per liter at 49° C. As concentrated solution made as can be separated from ore. Above sp. gr. 1.2 separation difficult; requires pressure filter.	Add NH ₄ OH. Mn(OH) ₂ precipitated on agitating with air as more easily filtered than Mn(OH) ₂ . Boil (NH ₄) ₂ SO ₄ solution with CaO. Absorb NH ₃ in fresh solution. Discard CaSO ₄ .	A conveniently strong pregnant solution made that is fairly readily separable from ore.	SO ₂ from furnace dissolved in water, then pure SO ₂ expelled by steam for dissolution of manganese as sulphite.
Precipitation; regeneration of precipitant	Solution evaporated to dryness (In spray drier). Nearly dry MnSO ₄ removed.	Add NH ₄ OH. Mn(OH) ₂ precipitated on agitating with air as more easily filtered than Mn(OH) ₂ . Boil (NH ₄) ₂ SO ₄ solution with CaO. Absorb NH ₃ in fresh solution. Discard CaSO ₄ .	Add Ca(OH) ₂ plus a little NaOH to keep OH ⁻ concentration high. Agitate with air. Regenerated CaCl ₂ returned to fresh batch ore; MnO ₂ precipitated.	Heat MnSO ₃ solution to expel SO ₂ and precipitate Mn(OH) ₃ or MnSO ₃ ; reuse SO ₂ . In experiments some MnSO ₄ was formed.
Treatment of precipitate	Calcine MnSO ₄ to Mn _x O _y and SO ₂ and SO ₃ .	Calcine Mn(OH) ₃ to Mn _x O _y .	MnO ₂ carefully - dry.	Mn(OH) ₃ or MnSO ₃ calcined to Mn _x O _y .
Regeneration of solvent	Return SO ₂ to re-leach fresh batch. Return SO ₃ .	No regeneration of solvent. Both the Ca and SO ₂ added discarded to waste as CaSO ₄ .		Return SO ₂ to leach fresh batch or ore.

No. 4. Roast with producer gas to reduce manganese minerals to MnO , leach with SO_2 out of the presence of air or oxygen (the SO_2 in furnace gas would need to be absorbed in water and expelled by steam as used) to obtain manganese as sulphite. Heat pregnant solution to expel SO_2 and thus precipitate manganese as sulphite or hydroxide, reuse the SO_2 and calcine the manganese product.

It is not contended that any methods proposed are practical under the present general conditions. Special local conditions that supply the needed chemicals at nominal cost or offer a market for a particular product may present an opening as a profitable operation.

The four methods are compared in table 69.

RECOVERY OF MANGANESE FROM SULPHATE SOLUTION

A cheap method for the recovery of manganese from the sulphate solution would very much aid the hydrometallurgy. This problem has been given much thought with generally negative results.

ELECTROLYSIS OF MANGANESE SULPHATE SOLUTION

The usual electrolysis failed because the manganese that deposited on the cathode readily redissolved in the acid formed. Attempts to protect the manganese by simultaneously depositing iron with the manganese on the cathode did not aid appreciably. Our experiments show that when an electric current is passed through a solution of manganese sulphate metallic manganese tends to deposit on the cathode, and more or less manganese dioxide is formed at the anode. It is, however, exceptionally difficult to obtain a high current efficiency at the cathode and to produce a satisfactory metallic deposit. A high current efficiency of MnO_2 formation at the anode is readily obtained.

A diaphragm seems to be necessary to separate the anode and cathode. The solution surrounding the cathode tends to become alkaline, and unless this solution is slightly acid manganese hydrate is deposited upon the cathode. On the other hand, the metal deposit is readily dissolved by an acid solution unless sufficient potential and current density are maintained at the cathode. For good results the solution must be exceptionally pure, and the concentration of acid and manganese sulphate in the catholyte must be maintained at a definite level. The presence of ammonium sulphate appears to be beneficial.

The best laboratory results were obtained as follows: A rotating nickel anode was enclosed in a cylindrical alundum diaphragm and surrounded by a cylindrical platinum anode. The catholyte contained 5.5 percent manganese as sulphate, 8 percent ammonium sulphate, and 0.04 percent free sulphuric acid. The acidity was maintained by supplying fresh solution containing 0.21 percent free acid at a rate sufficient to compensate for the electrolytic migration of the acid radical. The current density was 100 amperes per square foot of cathode area. A potential of 5 volts was required; the temperature of the electrolyte was $25^\circ C$. Under these conditions a good cathode deposit was

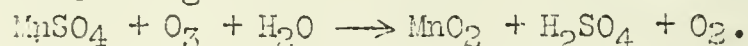
obtained, and the weight of metal formed indicated a current efficiency of 72 percent. If the electrolysis is continued much over an hour, however, some of the metal is apt to scale and drop off the cathode and redissolve in the acid electrolyte. This difficulty was not overcome, partly because of the difficulty in maintaining exactly the proper acidity in the catholyte.

Manganese dioxide is usually formed at the anode. The nature and amount of the dioxide deposit depend primarily on the kind of anode used and secondarily on the current density, temperature, and composition of the anolyte. High current density, high temperature, and high acidity favor the formation of manganese dioxide. On a platinum anode a dense, lustrous crystalline variety of dioxide is deposited. On anodes of other material, including graphite, carbon, lead, and manganese dioxide, the dioxide produced is usually amorphous.

To summarize; under suitable conditions metallic manganese can be electrolytically deposited from manganese sulphate solution, with or without simultaneous production of considerable manganese dioxide at the anode. No procedure was found, however, by which the metal could be retained and preserved on the cathode during an operating period greatly exceeding 1 hour.

PRECIPITATION OF MANGANESE DIOXIDE OZONE

Ozonized air, passed through a solution of manganese sulphate precipitates manganese dioxide according to the reaction



With adequate dispersion of the ozonized air in the solution, high utilization of the ozone is possible, but the cost of producing ozone was found to be so high that there is little prospect of using the method commercially to produce manganese dioxide.

BYPRODUCT PROCESSES

French Process

A. G. French has taken out United States Patents 1055157-8 under date of March 4, 1913. The outstanding feature in his process is the recovery of manganese dioxide from the sulphate solution by electrolysis. He proposes roasting sulphide ores containing manganese and zinc at 700° C., then mixing the calcine with 5 percent sodium bisulphate, grinding in water, and leaching to obtain the sulphates of manganese and zinc. After some purification this solution is electrolyzed between anodes of lead and cathodes of zinc. As the zinc is deposited on the cathode a corresponding amount of manganese is deposited at the anode as the dioxide. This could be made a source of considerable manganese dioxide, except that the product contains enough lead to prevent its use in the usual dry-cell battery and it has found very little sale in the chemical or paint industry. Our experiments show that part and probably most of the lead that appears as an impurity in this manganese mud was introduced through decomposition of the lead anode.

The only present use of this manganese dioxide mud is for oxidizing iron, arsenic, and antimony, so they are precipitated by hydrolysis from the leaching solutions in the electrolytic zinc process. Ralston (311) gives an excellent review covering recent litigation for the manufacture of manganese dioxide in electrolytic zinc practice.

Weldon Process

As early as 1866 the utilization of manganese was associated with the chemical industry, when manganese dioxide was employed in the manufacture of chlorine. The recovery of the manganese from the byproduct, the chloride of manganese, is known as the "Weldon process." Like most inventions, it is largely founded on previous experience and has been perfected by successive improvements. The waste still liquors are neutralized with limestone, and the milk of lime is then added in excess to precipitate all the manganese as the hydrate. The mixture is agitated with air, the reaction is rapid and heat-producing, and more manganous chloride is added to reduce the proportion or excess of quicklime; the resultant dark-brown precipitate, known as "Weldon mud," is mostly manganite of calcium (MnO_2CaO). The washed mud is pumped back to the chlorine generator, in which its reuse is similar in action to that of the original manganese dioxide and quicklime. A complete description, with a diagram of the apparatus used, is given in Thorpe's Chemical Dictionary, pages 13-18.

Various flotation experiments made by the author with the thought of removing the impurities from Weldon mud were unsuccessful. The use of barium hydroxide in the place of calcium hydroxide for precipitation of manganese from the sulphate solution gives a product from which the manganite of barium can be largely separated from the barium sulphate by careful hydraulic classification.

Other Methods Recorded by Patents

United States Patent 1415395 was granted to Chas. J. Reed, May 19, 1922, covering a process of making manganese dioxide from the sulphate. He proposes fusing manganese sulphate containing 2 or more molecules of water with just enough sodium nitrate to combine with the sulphate radical of the manganese sulphate. During the fusion the volatile oxides of nitrogen are expelled and recovered by condensation as byproduct or for reuse. The solid products of the reaction comprising manganese dioxide and sodium sulphate are discharged hot. After cooling the mass is broken up and leached with water. The sodium sulphate dissolves, leaving the manganese dioxide as the final residue.

United States Patent 1770791 was granted to Geo. N. Lebbby and Geo. D. Knight, July 15, 1930, for a sulphur dioxide method of leaching manganese ores, including a nitrate process for the recovery of manganese dioxide from the manganese sulphate. It appears to be intended as an improvement on the Westling process as used near Redwood City, Calif. They propose digesting the fine manganese ore in a series of three tanks, passing SO_2 gas through each tank in the series. As each tank is successively charged with enough SO_2 to dissolve

the manganese it is shut out of the circuit. Magnesium carbonate in the form of magnesite is added, and by the aid of air blown through the charge the iron and phosphorus are precipitated. At this stage in the operation, enough magnesium nitrate is added to react with the manganese sulphate; the charge then goes to filtration. If the filtrate contains copper it is precipitated by aluminum dust. The clean solution flows to the first evaporator, where it is concentrated to a specific gravity of 1.6. The concentrated solution is pumped to a retort and there heated from 200° to 500° C. until the manganese sulphate and magnesium nitrate react and form magnesium sulphate and manganese nitrate. The manganese nitrate is in turn converted into manganese dioxide and nitrous gases. The nitrous gases are oxidized by air to nitric oxide before the gas is passed to the condensing towers. The towers are filled with lump magnesite, with which the nitric acid reacts to form magnesium nitrate for reuse. The hot solution charge from the retort is filtered and washed to remove magnesium sulphate, which may be saved as a byproduct (epsom salts), if the market warrants the extra expense. The manganese dioxide residue remaining on the filter is the final product. The well-recognized disadvantage in this process, in addition to the cost of mechanical operation, is the excessive cost of nitrates. In the handling of the relatively large quantity of nitrates needed to react with the manganese extracted from the ore it has not been practical to prevent losses of nitrogen that have precluded the commercial use of nitric acid processes.

United States Patents 1266731-2 were issued to Geo. C. Westby, May 21, 1918. Each patent covers mechanical equipment for a proposed SO₂ leaching process. In the first patent he proposed to crush the ore coarse (about 1-inch), hydraulically separating the slime and treating the fines by spraying the pulp down a series of towers, countercurrent with SO₂ gas flow; the coarse ore was charged into percolation tanks with tight tops, the interstices between the ore lumps allowing easy upward passage of SO₂ gas through the charge. A fine spray of water or leach solution was added on top in quantity only sufficient to trickle through the charge and exit at the bottom. The second patent provided for a multiple series of shelves set staggered and filling the interior of a rectangular tank, the idea being to cover the shelves with fine ore charged at the top and to make the ore pass downward on the many shelves by vibration of each set. Steam and SO₂ were passed upward from the bottom and condensed leach liquor trickled downward through the charged apparatus. Both methods were tried at Ludwig, Nev., for leaching oxidized copper ores and were not commercially successful. The proposed methods offer less chance for success with the treatment of manganese ores.

United States Patent 1276739, as issued to C. Ellis, August 27, 1918, provides for leaching oxide manganese ores with acid-sodium sulphate to obtain manganese sulphate. His patent claim reads:

A process for making manganese dioxide which comprises reacting on raw material containing basic manganese compounds with acid material comprising acid sodium sulphate at reacting temperature to form manganese sulphate and converting of this salt into manganese dioxide.

G. D. van Arsdale and C. G. Maier obtained United States Patent 1304222 on May 20, 1919, for extracting manganese and making sulphuric acid and manganese dioxide. It consists in suspending ground manganese ore in water, adding acid to dissolve oxides of manganese other than dioxide, agitating and introducing sulphur dioxide gas into the pulp, and continuing to agitate while still conducting sulphur dioxide gas into the pulp until substantial completion of the reaction between the sulphur dioxide and the manganese compound contained in the ore, separating the resulting solution containing manganese from the leached ore residue, removing undesirable constituents of the solution, heating the manganese solution, and subjecting the heated manganese solution to electrolysis, making this electrolysis discontinuous as to the anode so as to produce desirable physical qualities in the separated manganese dioxide, continuing the electrolysis to attain a sufficient degree of purity and strength of the sulphuric acid produced and to obtain as dioxide enough of the manganese originally present, removing the substantially manganese free sulphuric acid thus produced, and in then removing the precipitated manganese dioxide from the anode.

The E. W. Haslup and B. A. Peacock process appears in United States Patents 1279108-10 of September 17, 1918, and 1291867 of January 21, 1919. It consists of treating a manganese ore with soluble sulphates, such as ferrous sulphate or aluminum sulphate, along with sulphuric acid of a gravity not more than 55° B., confining the ore mixture, devoid of added free water, for 15 hours or more in a closed receptacle from which the heat of reaction is prevented from escaping until a temperature of 150° C. is reached and until the desired values have been transformed into sulphates, and then extracting the sulphates with water.

C. S. Vadner, under United States Patent 1188705, of June 27, 1916, and 1236236, of August 7, 1917, claims a process for the recovery of manganese and zinc from oxidized or roasted sulphide ores. He proposed spraying the pulp into sulphur dioxide gas for dissolution as sulphates. The resultant solution is oxidized by air, powdered limestone is added to precipitate the iron, then at elevated temperature fresh limestone is added to precipitate the zinc, and finally by the use of quicklime and air the manganese is precipitated as weldon mud. A latter improvement provides for passing the pregnant solution over manganese dioxide ore to precipitate the iron. Part of the large excess of readily soluble manganese united with all the remaining acid, and in this neutral solution iron is not held in solution. Vadner also proposes to leach the SO₂ residue with brine (sodium chloride) to dissolve the lead.

Old Patents

Prof. E. H. Hersam gives the following review of old foreign processes for the treatment of manganese ores. They are of particular importance in showing the early development of hydrometallurgy of manganese.

In 1862 a process of ore treatment was patented in England by J. B. Readman, which consisted in fine-grinding the ore, mixing with sulphuric acid and sulphate of soda, drying, fusing, next adding lime and salt or bauxite, further igniting, and finally extracting with hot water, precipitating the sulphides of nickel and cobalt if present by sodium sulphide, evaporating the solution to crystallize the sodium sulphate, evaporating the mother liquor to dryness, and igniting in a muffle furnace with charcoal to produce the sulphide of manganese.

The Herrenschmidt and Constable process (British Patent 1535), which followed the above-described process, was devised in Australia. The treatment proposed, consisted in applying to the ores either warmed or at the ordinary temperature, a solution of the sulphate of iron which, as applied, reacts with oxides of manganese, and cobalt of the ore, produces the sulphate of these metals, and deposits the oxide of iron.

In 1884, appeared Readman's second English patent for a process of treating manganese ores. The process consists in pulverizing the ore, treating it with a solution of soluble chloride such as that of sodium, calcium, magnesium or iron, agitating with or without heat, and effecting the mixing in a furnace if desired. For the extraction of cobalt, iron chloride is preferred to the other chlorides, the desired amount being somewhat more than 1 part of the anhydrous chloride of iron to 1 part of ore. The mass, in any case, is heated to a temperature sufficient to decompose the iron chloride. The mass is then lixiviated with water to extract cobalt, nickel, and manganese and the very small amount of iron that dissolves.

A German patent (May 29, 1884) succeeding the one above-mentioned, was granted to R. Nithack for the injection of a spray of concentrated solution of the chlorides of these metals upon hot, inclined, fire clay tiles in an appliance provided for regenerating the chloride. By the action thus claimed the metal takes an oxygen, becomes a protoxide, and releases its chlorine, which is recovered in the form of hydrochloric acid for the further extraction.

In September of 1884, a patent was granted in England (No. 12044) to Herrenschmidt for an extension of his original process. The process aims chiefly towards the extraction of nickel but includes manganese and cobalt. For ore containing manganese, the material is ground and lixiviated hot with ferrous chloride, which results in the decomposition of the nickel, cobalt, and manganese minerals. The nickel and cobalt are precipitated from the solution by adding manganese sulphide or the hydrated oxide of manganese. The pure solution is either crystallized or evaporated to dryness to obtain hydrochloric acid and manganese oxide.

Further improvements in the Herrenschmidt and Constable process appeared in 1885 (Dingler's polyt. Jour., vol. 252, p. 392), in which it was shown that, in the treatment of manganese and cobalt ores, it is best to boil with ferrous sulphate solution for a half hour, after which the clear solution may be decanted and the protosulphate of the metal oxidized to higher salts.

By a process devised by Campbell and Boyd (British patent 12426, June 24, 1893), the sulphate of manganese, in solution, obtained as a by-product in the manufacture of chlorine, is converted into a state of carbonate suitable for producing oxide by the Dunlop process. In this process the solution of sulphate is mixed with a solution of sodium carbonate, and the precipitated manganese carbonate is collected, washed, and treated by the Dunlop process for the production of the oxide. A modification of this method consists in running a manganese solution slowly into a solution of caustic soda, blowing air into the mixture the while, and continuing aeration for a time after neutrality is attained.

By a patented process of Albright and Hood (British patent 9271, May 1893), manganese ore is mixed with coal dust and sulphuric acid, after which it is heated on a furnace bed to produce manganese sulphate, which then is leached from the digested product. The solution of manganous sulphate is treated with calcium chloride, in excess, after which the precipitated calcium sulphate is separated. The washings are brought to a high state of concentration of the calcium chloride, lime is added and the solution is then treated with air in a Weldon oxidizer to produce the Weldon mud, a product before described. Another means of effecting this precipitation, that was suggested by these authors, was the use of ammonia as the precipitating agent. The excess of ammonia is recovered at the time of the oxidation of the precipitated manganese hydrate, and as sulphate by evaporation.

In 1894, A. R. Davis of Manchester patented (British patent 2696) a process of treating manganese peroxide with ferrous sulphate and separating the soluble sulphate of manganese from the iron thereby formed. The sulphate of manganese was to be used as such or to be evaporated for the production of the solid sulphate of manganese.

In 1895, Greene and Wahl (Dingler's polyt. Jour., 1895, vol. 296, pp. 112-114) prepared pure manganese, freed from iron, by crushing the ore, leaching the ore down to 0.1 percent with sulphuric acid in 2 to 3 volumes of water. By the treatment, the iron is extracted and the manganese dioxide remains unattacked. Their process included an extension of the treatment for the production of manganese free from carbon and iron, which consists in igniting the dioxide of manganese, heating to 400° to 500° C. to form Mn_3O_4 , further reducing by the introduction of volatile hydrocarbons and finally in crucibles with aluminum or magnesium.

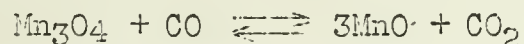
CHAPTER 8. PYROMETALLURGICAL TREATMENT OF MANGANESE ORES

By T. L. Joseph⁴

PRODUCTION OF FERROMANGANESE

Reduction of Manganese Oxides

According to Meyer and Rötgers (256) heat alone will dissociate MnO_2 into Mn_2O_3 at 530°C. at atmospheric pressure. Manganic oxide (Mn_2O_3) is further dissociated into Mn_2O_4 at 940°C. under atmospheric pressure. Dissociation by heat alone is unimportant in blast-furnace practice, because reduction of the higher oxides occurs at temperatures below those required for thermal dissociation. Clevenger and Caron (50) found that the MnO_2 in a refractory silver ore was reduced to MnO in 2 hours at 465°C. in the presence of dry carbon monoxide. Nishibori (275) investigated the equilibrium conditions for the reaction



at 780°C. During the early stages of reduction virtually all of the carbon monoxide was converted to carbon dioxide. As the composition of the solid phase approached MnO the CO_2 in the gas phase decreased sharply from 100 to about 7 percent. These data show that gas with a $\text{CO}:\text{CO}_2$ ratio of 14:1, will reduce Mn_3O_4 to MnO at 780°C.

Reduction of Manganous Oxide

Several investigators have determined the temperature at which reduction of manganous oxide begins. Wüst (399) has reported the work of Meyer, who found that carbon monoxide will reduce this oxide of manganese in the presence of metallic iron at $1,150^\circ \text{C.}$ Precautions were taken during reduction with CO to prevent carbon deposition. Reduction in the presence of solid carbon occurred at $1,100^\circ \text{C.}$ Greenwood (124) has reported that reduction by retort carbon in the presence of electrolytic iron began at $1,030^\circ \text{C.}$ and in the absence of iron at $1,105^\circ \text{C.}$ He refers to metallic globules and to partial fusion of the metallic residue, which he separated from the unreduced oxide by treatment with sulphuric acid. Since metallic manganese melts at $1,244^\circ \text{C.},$ it appears that his observed temperatures were lower than the true temperature. Royster (322, p. 119) states that laboratory tests strongly indicate that $1,265^\circ \text{C.}$ is the lowest temperature at which MnO is reducible by carbon or carbon monoxide. The results of a single experiment reported by Nishibori (275, p. 296) indicate that reduction of Mn_3O_4 will progress beyond MnO at 780°C. in the presence of a gas phase consisting of 93.5 percent CO and 6.5 percent CO_2 . The strong reducing action of the product of reduction towards sulphuric acid, a property not characteristic of MnO , was attributed to the presence of manganese or a suboxide of manganese.

⁴ Metallurgist, U. S. Bureau of Mines.

The results of these various investigators indicate that temperatures of 1,100° to 1,265° C. are necessary to reduce appreciable amounts of MnO in the presence of solid carbon and carbon monoxide.

Factors Controlling Amount of Carbon in Liquid
Manganese Produced in the Presence of Carbon

Moissan (262) reduced Mn_3O_4 in the electric furnace in the presence of varying amounts of carbon. The fused metal produced in four experiments varied in percentage composition as follows:

	<u>1</u>	<u>2</u>	<u>3</u>	<u>4</u>
Mn	85.0	85.82	90.6	94.06
C	14.59	13.98	10.2	6.35

This variation in carbon content was attributed to the relative amounts of manganous oxide and carbon in the charge and to the temperature. In the presence of excess oxide fusions containing 4 to 5 percent carbon were obtained. When reduction was carried out at low temperatures, Moissan obtained metal as follows:

	<u>1</u>	<u>2</u>
Mn per cent	93.2	96.2
C do	4.5	3.6

Roberts and Wraight (316, p. 241) have also observed the effect of temperature upon the amount of carbon absorbed during reduction. They heated mixtures of pure Mn_3O_4 and powdered wood charcoal. When the estimated temperatures of reduction varied from 1,600° to 1,770° C. the proportion of carbon in the metal increased from 5.5 to 7.44 percent. Doubling the amount of reducing agent had little effect upon the amount of carbon absorbed by the metal.

Three tests were made by W. F. Holbrook and the writer to observe the amount of carbon that will dissolve manganese. In the first test a charge of C. P. manganese carbonate and finely divided charcoal was gradually heated to 1,900° C. Due to interference from fumes this temperature, observed with an optical pyrometer, is probably lower than the true temperature. The metal obtained consisted of small shot distributed throughout a mass of charcoal. Their carbon content ranged from 7.24 to 8.81 percent.

A smaller amount of charcoal was used in the second test, from which a slug of metal was obtained. The peak temperature observed was 1,900° C. Carbon determinations on portions of the piece of metal obtained varied from 7.97 to 8.52, averaging 8.22.

In the third test 30 grams of metal obtained from the two previous tests were heated in a graphite container closed at the top with a cone-shaped graphite rod. The object of this device was to hold the bath under some

pressure, so that higher temperatures could be reached. During 47 minutes the temperature was gradually raised from 1,320° to 2,000° C. A clearer atmosphere in the furnace permitted more satisfactory readings than in the first two tests. At the end of the heating period the graphite plug was removed, and the graphite container was set in a steel cylinder surrounded by ice water. At the instant the graphite plug or cover was removed the metal boiled vigorously. A small round ingot, 5/8 inch by about 1 inch and weighing 27 grams, was recovered and sectioned for analysis. Various sections of the ingot contained the following percentages of carbon: 9.85, 10.00, 10.14, and 10.75, averaging 10.19.

Graphitic carbon determinations averaged 3.31 percent, leaving 6.88 percent combined carbon, which checks with the theoretical amount corresponding to the formula Mn_2C . It appears that the 3.31 percent graphitic carbon was either in solution as such or that the compound Mn_2C , which would correspond to about 10 percent carbon, broke down upon cooling.

Near the top of the ingot an irregular section was found to contain 11.36 percent carbon. This higher manganese content is attributed to contamination of the sample with graphite floating upon the surface of the melt. One could continue heating a melt of manganese until nothing but a residue of carbon remained; that is, as manganese is volatilized there might be an enrichment of mechanically held graphite. Such a condition could not account for the 3.31 percent graphitic carbon found in the third melt, because about 88 percent of the manganese charged was recovered.

Since Moissan's experiments were presumably conducted at about atmospheric pressure the manganese content (14.59 percent) appears to be due either to carburization in the solid state or to contamination of samples by a residue of mechanically held graphite. Whatever the mechanism of reduction may be, it appears that the absorption of carbon by liquid manganese is a function of temperature.

The experiments of Ruff and Bormann (328) indicate that a homogeneous liquid phase containing more than 7.12 percent manganese cannot exist at 30 mm pressure. Under this pressure the solution boiled at 1,526° C. Their work does not, however, indicate the amount of carbon that can be held in solution by liquid manganese at higher temperatures attainable at higher pressures.

Refining Action of Manganous Oxide upon Ferromanganese

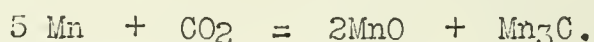
The refining or decarburizing action of a slag rich in MnO has been studied by a number of investigators in connection with the problem of producing low-carbon ferromanganese, which is discussed on page 218. Roberts and Wraight (316, p. 281) reduced the carbon content of ferromanganese from 6.72 to 2.79 percent by heating it 3 hours under a semifused pasty mass of MnO at temperatures above 1,600° C. Humbert (173) has patented a method for reducing the carbon content of ferromanganese to about 1 percent or less by heating in the presence of manganous oxide at about 1,700° C. A similar method for decarburizing ferromanganese has been patented by Hadfield (134).

Experimental evidence is not available to show definitely whether in high-temperature reduction liquid-manganese carbides are first formed and undergo subsequent refining in the presence of MnO or whether liquid metallic manganese is first formed and absorbs carbon in amounts depending upon the temperature and presence of oxides. Both reactions probably occur in the blast furnace.

Carburization of Manganese Produced in the Solid State

The reduction of MnO at temperatures below the melting point of manganese has received comparatively little attention. Reference has been made to Greenwood's (124) tests at reported temperatures of 1,030° to 1,105° C. The metallic residue, after separation from excess MnO, was treated with hydrochloric acid. From the composition of the gas evolved the carbon content was calculated to be 6 percent. The metallic residue had undergone partial fusion so that the carbon content may have been due to the combined effects of solution and the well-known reaction of case carburizing.

Muller and Barck (266) introduced a mixture of 43.3 cm³ of carbon dioxide and 43.8 cm³ of hydrogen over a sample of pulverized manganese and held the system at 800° C. until no further change in the volume of gas occurred (2 hours). The residual gas proved to be pure hydrogen and occupied a volume of 43.6 cm³. The solid phase was identified as MnO and Mn₃C. The net result of the reaction may be expressed by the following equation



Hilpert and Paunescu (163, p. 3479) heated finely divided manganese in a stream of methane and found that about 6 percent carbon was absorbed in an hour at 900° C. After 10 to 11 hours exposure at 900° C. the metal absorbed as much as 15 percent carbon. Carburization was much more rapid at 800° and at 900° C. than at lower temperatures less favorable to dissociation of methane into carbon and hydrogen. It is evident that under some conditions finely divided manganese carburizes very readily. In view of the readiness with which manganese carburizes it seems likely that if solid manganese is formed it would be converted very quickly by either CO or solid carbon to manganese carbide.

Reduction of Manganous Oxide in the Crucible of the Ferromanganese Furnace

One objective of good operating practice is to reduce the manganese in the slag to the lowest point consistent with fuel consumption and with tonnages that give the most economical operation. Crucible reactions are important because the manganese content of the slag tapped will depend upon the direction in which the crucible reactions are proceeding.

According to Wüst and Meyer (400), no reduction of manganese can occur below the tuyeres in pig-iron furnaces because the slag is oxidizing toward the metal. Manganese equilibrium constants obtained from steel-making data were used to determine whether the slag tapped from a large number of pig-iron furnaces was oxidizing or reducing towards the metal produced. The manganese equilibrium values $K_{\text{Mn}} = \frac{\text{MnO (slag) times Fe (metal)}}{\text{FeO (slag) times Mn (metal)}}$, calculated from

published blast-furnace data, were much smaller than values of K_{Mn} obtained from steel-making data. They concluded from this that slag from pig-iron furnaces is normally oxidizing toward the metal. Under such conditions no reduction of MnO , but rather oxidation of manganese, would occur below the tuyères if slag-metal reactions predominate over reduction of manganous oxide by carbon or by carbon monoxide.

The conclusion that manganese cannot be reduced below the tuyères of a pig-iron furnace is based upon three major assumptions: (1) Slag-metal reactions control the reduction of manganous oxide in the blast-furnace crucible in which there is a large excess of carbon existing as coke and carbon monoxide; (2) values of manganese equilibrium constants, obtained from steel-making data, can be applied rigorously to the crucible of a blast furnace; and (3) FeO normally reported in blast-furnace slag exists as such and not as metallic iron.

To determine whether manganese was reduced from slag which was oxidizing toward the metal the progress of the reduction of manganese from a mixture proportioned to produce ferromanganese and held at $1,550^{\circ} C.$ was followed by a series of slag and metal samples. A charge of 275 grams of manganese ore, iron ore, and sufficient quantities of lime, alumina, and silica to produce about equal amounts of slag and metal was melted as rapidly as possible in a graphite crucible heated in an induction furnace. The melt was poured, cooled, and transferred to another graphite crucible for the reduction test. Charcoal was added as a reducing agent. The manganese and iron content of eight samples of slag and of metal, taken over a period of 2 hours after the slag was fused, are given in table 70. The same data are shown graphically in figure 26.

The small values of K_{Mn} , given in table 70, show that slag was very oxidizing to the metal throughout the whole period. Notwithstanding this fact, gradual reduction of manganous oxide occurred. The first metal sample contained 66.4 percent manganese and 25.85 percent iron. This metal was produced at $1,560^{\circ} C.$ By applying the value of $K_{Mn} = 240$, reported by Korber (211) for this temperature, the ratio of MnO to FeO which must exist in the slag before reduction of manganese could occur by slag-metal reactions may be found as follows:

$$240 = \frac{MnO}{FeO} \times \frac{25.85}{66.4},$$

$$\text{or} \quad \frac{MnO}{FeO} = \frac{66.4}{25.85} \times 240 = 616.$$

Thus it may be seen that to produce metal containing 66.4 percent manganese by slag-metal reactions at $1,560^{\circ} C.$ it would be necessary to have more than 600 times as much MnO as FeO in the slag. The production of such metal over a period when the ratio of MnO to FeO ranged from about 16 to 25 indicates the reaction between MnO in the slag and carbon or carbon monoxide, rather than slag-metal reactions, controls the reduction of manganous oxide under highly reducing conditions.

TABLE 70. - Manganese and Iron in Slag and in Metal
During Ferromanganese Reduction Test

Sample No.	Time after charge melted, minutes	Slag, percent		Metal, percent		$K_{Mn} = \frac{MnO \text{ (slag)} \times Fe \text{ (metal)}}{FeO \text{ (slag)} \times Mn \text{ (metal)}}$
		MnO	FeO	Mn	Fe	
1	0	60.0	3.68
2	20	55.5	2.00	66.4	25.85	11.0
3	30	50.2	1.17	72.6	20.6	12.0
4	40	50.2	0.94	75.2	17.4	13.0
5	60	39.9	.25	81.0	11.8	23.0
6	75	30.0	.21	83.3	9.5	16.3
7	105	17.2	.36	83.8	9.0	4.5
Tap	125	11.25	.10	84.9	8.28	11.0

Tap slag, percent: SiO₂, 32.2; Al₂O₃, 9.05; CaO, 41.04.

Unless special precautions are taken to free slag samples from metallic shot, total iron determinations converted to a FeO basis have little significance. The slag samples reported in table 70 were cleaned of metal by grinding the slag and separating it into the following screen sizes: -100 +120, -120 +140, -140 +160, -160 +180, -180 +200, -200 +270, -270 +400, -400. Each of the foregoing screen fractions was given a wet gravity separation. The heads from each sample were treated with a solution of copper sulphate and examined under the microscope before acceptance for analysis. Metal particles were copper-plated and therefore readily distinguished from the slag. In doubtful cases a microscopic count of the metal particles was made to be certain that contamination of the slag with iron from the metal would not exceed a few hundredths of 1 percent.

The experience gained in freeing these samples from metal leads one to believe that much of the iron reported in routine samples of blast-furnace slag occurs as metallic iron and not as FeO. It appears likely that the slags from pig-iron furnaces, considered by Wüst and Meyer, contained considerable metallic iron which was reported as FeO.

Another crucible test was made with a 300-gram charge to study the relative importance of slag-metal reactions and reduction of MnO by C or CO in the production of metal in the spiegeleisen range. The charge contained about three times as much manganese as iron and amounts of lime, alumina, and silica to yield a final slag of about the same composition as that produced in the ferromanganese reduction test.

Manganese equilibrium constants computed from the composition of metal shot and the slag from which they were recovered show that manganese was formed in the slag bath when the slag was oxidizing toward the metal. The data from this test are given in table 71.

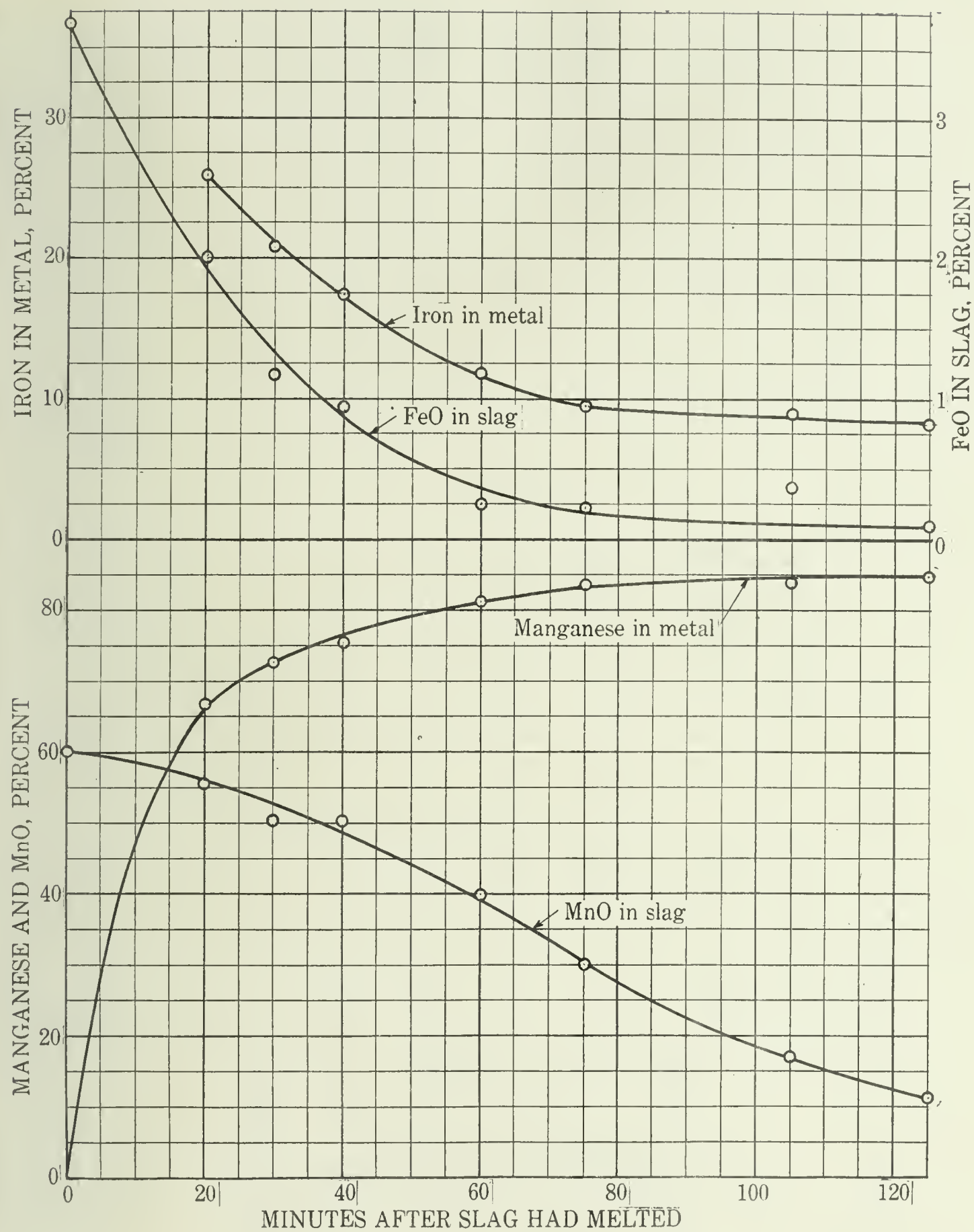


Figure 26.— Changes in slag and metal during ferromanganese reduction test.

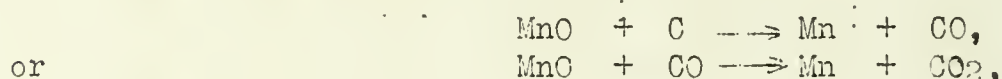
TABLE 71. - Changes in Composition of Slag and Metal During Spiegeleisen Reduction Test

Sample no.	Time after slag had melted, minutes	Metal shot removed from slag, percent		Bath metal, percent		Slag, percent		$K_{Mn} = \frac{MnO (slag)}{FeO (slag)} \times \frac{Fe (metal shot)}{Mn (metal shot)}$	Temperature, °C.
		Mn	Fe	Mn	Fe	MnO	FeO		
1	0	29.2	9.32	...	1,553
2	20	15.0	81.0	5.13	89.0	33.0	1.57	110	1,535
3	21 (Added 5 g. Fe_2O_3)	10.90	83.0	29.1	.50	...	1,546
4	41 (Added 5 g. Fe_2O_3)	49.6	46.4	13.6	80.0	24.4	.41	56	1,535
5	60 61 (Added 10 g. Fe_2O_3)	49.5	46.5	21.7	.44	47	1,542
6	80 81 (Added 10 g. Fe_2O_3)	24.0	72.0	15.2	78.8	21.4	.49	131	1,530
7	109 (Added 10 g. Fe_2O_3)	23.6	72.4	23.9	2.29	32	1,535
8	114 130 138	16.0	78.0	
9	143 (Added 10 g. Fe_2O_3)	12.3	83.7	18.5	2.33	55	-
10	148	22.9	73.1	18.3	.89	65	-
11	153	21.6	74.4	16.4	.65	87	1,535
Tap	163	23.5	72.5	17.1	77.0	14.1	.31	139	-

Tap slag, percent: SiO_2 , 32.74; CaO , 42.3; Al_2O_3 , 9.64.

It is well-known that the first metal produced from a mixture of the oxides of iron and manganese is higher in iron than metal produced later. Metal shot picked from the slag samples were used therefore to ascertain the composition of the metal produced during the various stages of reduction. These shot were invariably higher in manganese than the bath metal. The analyses of the shot and the bath metal are given in table 71. Small metal shot do not settle rapidly from a slag bath. They represent metal produced when the iron oxide in the slag was higher than at the instant the slag samples were taken. The slag was therefore more oxidizing toward the metal than the manganese equilibrium constants in table 71 indicate. The gradual decrease of manganese in the slag and the increase of manganese in the bath metal show that manganous oxide was reduced throughout the test. Ferric oxide was added from time to time to maintain a concentration of iron oxide in the slag readily determinable analytically and readily distinguishable from small amounts of metallic iron.

According to the concept that slag-metal reactions control the reduction of manganous oxide from slag and assuming that manganese equilibrium constants obtained from steel-making data can be applied to a system containing excess C and CO, slag 7 in table 71 should have contained about 80 times as much MnO as FeO to permit the forming of metal containing 23.6 percent manganese. Metal shot containing this concentration of manganese were actually formed in a slag containing only about 10 times as much MnO as FeO. The data, as a whole, point very definitely to the conclusion that reduction of MnO from slag in the presence of an excess of CO and C is not determined by slag-metal reactions but by those which follow:



Concentrations of FeO necessary to oxidize manganese in the metal do not persist if the slag is fluid and a large excess of CO and C is present.

Wüst (399) observed that metal shot recovered from viscous slag were higher in manganese than the bath metal. He cited this result as evidence to support the view that manganous oxide is reduced above the tuyères and that no further reduction but oxidation of manganese occurs in the crucible below the tuyères. In this connection he states:

There might exist the possibility that during retention of the washed iron (metal shot) in the slag, the silicon, manganese, and the phosphorus are reduced out of the slag by the carbide carbon of the iron. This conception is, however, negatived by the fact that so long as iron oxide is present in the slag in considerable quantity, a reduction of the above-mentioned oxides can not take place.

The presence of gas containing 34 percent CO and fractions of 1 percent CO₂ and the presence of a large excess of coke are not taken into consideration in the foregoing statement. One would expect the metal bath containing the first metal produced to be lower in manganese than metal shot formed within the slag after it has been partly depleted of iron oxide. Lumps of

coke forced down into the slag bath below the inactive plastic core in the blast furnace no doubt act as nuclei around which metal reduced from the slag is formed. This phenomenon was shown by small particles of metal-coated charcoal found in samples taken during the crucible tests previously described.

Tests Showing Manganese is Reduced Below Tuyères. - The results from crucible tests indicating that reduction of manganous oxide below the tuyères is largely controlled by reaction with C or CO, and not by slag-metal reactions, is borne out by the manganese content of liquid metal samples removed from the crucible of pig-iron furnaces. Bansen (14) has reported average analyses of metal samples removed from the lower part of a pig-iron furnace as follows:

Sampling level	Composition of metal, percent			
	C	Si	Mn	P
6 ft. 9 in. above main tuyères	1.32	1.86	0.66	0.58
Main tuyère level	2.81	1.33	.86	1.03
33.5 inches below tuyères	4.06	2.03	1.55	1.79
Tapped metal	3.64	.45	1.58	1.80

The above analyses show that metal from the tuyère level contained about one half as much manganese as the tapped metal and metal removed from a slag notch. Analysis of partly reduced ore taken from the slag notch during this investigation indicates that manganese was reduced when the slag was very oxidizing towards the metal according to manganese equilibrium constants obtained from steel-making conditions. The reduction of manganous oxide under such conditions shows that, except for periods of hanging and slipping when unusually large amounts of unreduced oxides may be brought into the crucible, slag-metal reactions play a minor part in the reduction of manganous oxide below the tuyères.

Mund, Stoecker, and Eilender (267) removed 600 liquid samples and 150 gas samples from a blast furnace across four levels as follows: Auxiliary tuyères, 6 feet above the main tuyères; main tuyères; main slag notch, 3.2 feet above the hearth; and auxiliary slag notch, 4.5 feet above the hearth. The analyses of the metal samples taken were correlated with the analyses of 300 samples of pig iron. Samples were taken through plugged and through normally-working tuyères. The manganese content of the metal samples removed from the auxiliary tuyères was about two thirds that of the tapped metal. In front of both the auxiliary and main tuyères a strong oxidation of manganese occurred, particularly in the oxidizing portions of the combustion zones that extended about 32 inches beyond the nose of the tuyères. From about 32 to 48 inches from the nose of the tuyères the manganese increased rapidly and at the latter position approached the manganese content of the tapped pig iron. Beyond this position the manganese content decreased, presumably due to a decrease in the temperature prevailing in the central part of the idle core or what the German investigators have called the "stille Mann." It would appear that more metal moves down through the periphery than through the central inactive core. Samples drawn from sections of the periphery were therefore more representative

of the metal passing any particular plane. In view of the oxidation of manganese in the combustion zones, it is evident that a large amount of manganese was formed below the tuyères. The increase in the manganese in the metal between normally working tuyères and the slag notch also shows very clearly that a great deal of manganous oxide is reduced below the tuyères. Tapped metal was higher in manganese than that removed from the slag notch, indicating that a considerable amount of manganous oxide was reduced from the slag layer adjacent to the metal bath.

No direct information, similar to that obtained on pig-iron furnaces, is available on the conditions in a ferromanganese furnace. The gas samples removed by Perrott and Kinney (300) from the tuyères of a furnace producing ferromanganese show, however, a decided deficiency of oxygen up to a position about 32 inches from the nose of the tuyères. From the converter process for making steel one would expect that the major part of this oxygen is removed from the gas phase by the oxidation of the manganese which is again reduced in the crucible.

Inactive Zone in Bosh and Crucible. - A picture of the conditions existing in the crucible of a blast furnace is essential in obtaining an understanding of crucible reactions. The investigations of Ebelmen, Van Vloten, Perrott and Kinney, and Lennings have definitely established the existence of localized combustion zones in front of the tuyères. In formulating a conception of the effect of these localized zones upon stock flow in the furnace, it appeared to Royster and the writer (323) that a central area of the furnace at the tuyère level is comparatively inactive. Stoecker (357) concluded from explorations of the hearth and bosh that this inactive zone (or dead man) begins above the main tuyère level, at which it was roughly 6 feet in diameter. At this level it consists of unreduced ore and coke. Below the tuyères it widens and rests on the hearth. At the bottom it consists of coke, only. The dimensions of this inactive core and its permeability vary depending upon the diameter of the hearth and the amount of coke used per ton of iron.

In observing the rate of stock descent in various parts of the furnace, Kinney (206) observed a pendulumlike movement of the stock column during charging, indicating that the lower part of the stock column was supported by a fluid. If one considers the volume occupied by the metal and slag previous to a metal cast, it becomes obvious that only a part of the volume available below the tuyères is occupied by slag and metal. It appears therefore that a core of plastic material, extending below the tuyère, and coke, submerged in the liquid bath, occupy a considerable portion of the crucible. Whether the inactive core rests on the hearth would, it appears, depend upon the depth of the metal layer and the upward pressure exerted by the blast of air applied through the tuyères.

Kinney (207) observed a temperature of 1,250° C. in the center of a 300-ton foundry furnace having a hearth diameter of 14 feet 6 inches, compared to a tuyère temperature of 1,534° C. Rheinlander (313) has also observed that the center of the furnace is comparatively cold. He states that, although the temperature can not be rigidly established, about 4.5 feet beyond the tuyères it is under 1,500° C. In the central part of the core the temperature varied between 1,000° and 1,100° C.

Although there is no direct information about the existence of a comparatively cold inactive zone in the central part of the crucibles of ferromanganese furnaces it seems quite certain that such an idle core exists in some modified form. In the ferromanganese furnace, as in pig-iron furnaces, the major part of the metal passes through the oxidizing atmosphere in front of the tuyères. Manganous oxide so formed is reduced from the slag surrounding this inactive core and to some extent within the core itself. The low temperature in the central area does not, however, favor manganese reduction. Metal high in manganese may be found in this central core, but it is a minor part of the total metal produced. The final reduction of manganous oxide, which has a pronounced effect upon the manganese carried out in the slag, occurs in the slag layer, which contains coke forced into the slag by the overlying stock column. This layer of slag, in which the temperature is maintained by convection, is hotter than the central core through which heat is transmitted largely by conduction. Coke thus submerged in the slag forms strongly reducing areas. The coke is also effective in maintaining a gas phase consisting of about 34 percent CO and fractions of 1 percent CO₂, the balance being largely nitrogen.

Melting Points of Manganese Silicates

The extent to which MnO can be reduced without fusion depends upon its purity, particularly with respect to silica. Manganous oxide melts at approximately 1,600° C. and can therefore be reduced without fusion. Dorinckel (73) has studied the system MnO-SiO₂. According to his equilibrium diagram, tephroite (2MnO.SiO₂, 29.8 weight-percent SiO₂) solidified at 1,323° C. and rhodonite (MnO.SiO₂, 45.9 weight-percent SiO₂) freezes at 1,230° C. A eutectic between these two silicates, containing 41 percent SiO₂ by weight, solidifies at 1,190° C. Herty, Conley, and Royer (158), who have recently studied this same system, report somewhat higher melting temperatures for corresponding points in the equilibrium diagram.

Figure 27, from the report of Herty, Conley, and Royer, contains the ternary diagram for the system FeO.MnO.SiO₂ and the binary diagrams for the systems FeO.SiO₂, MnO.SiO₂, and MnO.FeO. For present consideration, the binary system MnO.SiO₂ and that portion of the ternary system adjacent to it have primary importance. The major part of the oxygen combined with the iron in manganese ores is probably removed before fusion temperatures are reached. Higher oxides of manganese are reduced to manganous oxide. By the time the particle reaches fusing temperatures manganous oxide and silica are the major constituents of ores in which the gangue is chiefly silica. Fusing temperatures will depend upon the proportions of silica and manganous oxide present and will range from about 1,600° C. for pure manganous oxide to 1,250° to 1,350° C. in the range of 30 to 40 percent silica.

Incomplete reduction of ferrous oxide (due to lack of time, size of ore particle, or low permeability to reducing gases) would tend to lower the melting points of the ore particle undergoing reduction. Herty, Conley, and Royer have determined the melting points of slags whose compositions are indicated by the circles in the ternary diagram of figure 27. A melting temperature of 1,320° C. has been reported for a slag containing 9.01 percent FeO,

83.6 percent MnO , and 7.39 percent SiO_2 . If such a three-component slag were converted to a two-component slag by reduction of the ferrous oxide the melting point would be increased to about $1,540^{\circ} C$. Ferrous oxide has less effect in lowering the melting temperature in the range of higher-silica content. If a three-component slag composed of 12.42 percent FeO , 72.74 percent MnO , and 14.84 percent SiO_2 , with a melting point of $1,310^{\circ} C$. is converted to a two-component slag by reduction of ferrous oxide, the melting point would increase to $1,475^{\circ} C$. Thus it may be seen that incomplete reduction of ferrous oxide would promote the formation of liquid silicates and tend to minimize reduction in the solid state.

Many commercial ores contain about 8 percent silica and 50 percent manganese. After the higher oxides of manganese and the iron minerals are reduced, the remaining portion of such material would contain about 90 percent manganous oxide and 10 percent silica and would have a melting temperature of $1,475^{\circ}$ to $1,500^{\circ} C$. Before such temperatures are reached, however, reduction of manganous oxide occurs along the periphery of the ore particles. As reduction proceeds the ratio of silica to manganese increases until it reaches a point at which the outer shell first softens and finally melts. The manganese silicates so produced subsequently combine with the calcium oxide and magnesium oxide. These basic oxides would be absorbed in the initial slag formed at points of direct contact between the lumps of ore and calcined limestone. The effect of the calcium oxide in retarding the formation of manganese silicates and in raising the free-running temperature of the slag tends to increase hearth temperatures.

A more siliceous manganese ore containing, for example, 15 percent silica and 45 percent manganese, would contain about 80 percent manganous oxide and 20 percent silica after the higher oxides of manganese and the iron minerals had been reduced. This material would melt at about $1,450^{\circ} C$. which is in the temperature range required for rapid reduction of manganous oxide. Reduction of comparatively small amounts of manganous oxide along the outer surface of the piece would concentrate the silica to a point that would permit softening and fusion. Thus it may be seen that silica in the ore promotes the formation of manganese silicates and tends to minimize the amount of manganous oxide reduced prior to fusion. After the slag is formed high silica in the charge results in a dilution of manganous oxide in the slag. In addition to the dilution due to silica, there is further dilution as a result of the calcium oxide used to flux the silica.

It appears from data obtained on pig-iron furnaces that some reduction of manganous oxide occurs before fusion. Part of the manganese reduced above the tuyères is oxidized in passing through the combustion zones. Final reduction in the crucible, which governs the loss of manganese in the slag, is very important. Adjustments in fuel requirements, in the blast temperature, in the speed of driving, and in slag basicity, together with the selection of raw materials of low-silica content are of foremost importance in minimizing the loss of manganese in the slag.

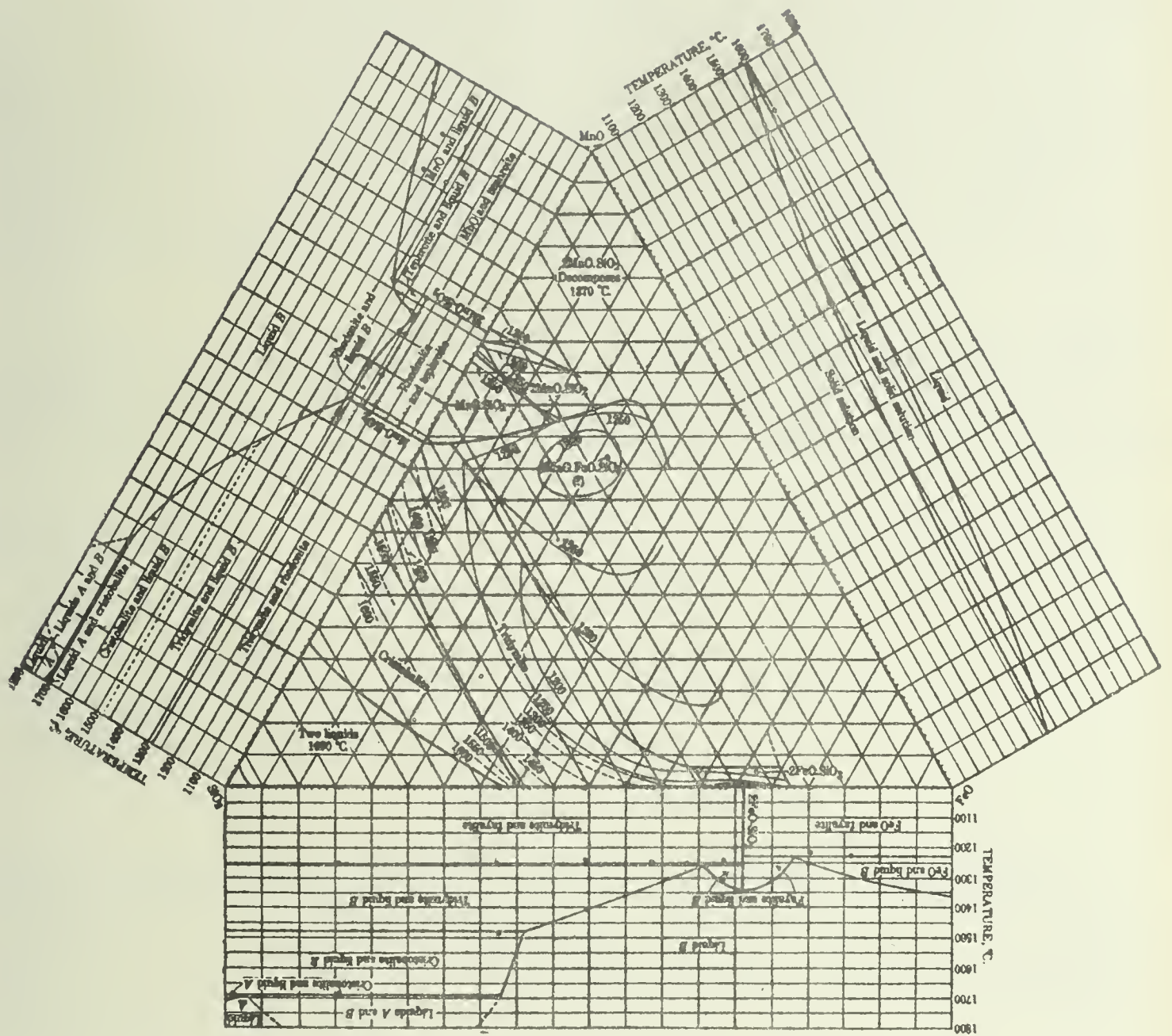


Figure 27.— Ternary diagram $\text{FeO}-\text{MnO}-\text{SiO}_2$ with binary systems.

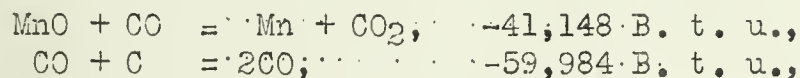
Reduction of Manganese from Slag

Christianson and Hunter (48) obtained data on the reduction of manganous oxide in an investigation of the Jones process. Samples of manganiferous iron ore (10 to 24 percent manganese, 24 to 40 percent iron, and 10 to 26 percent silica) were mixed with coal and heated in graphite crucibles for 30 to 40 minutes at 1,250° to 1,300° C. About 93 percent of the iron and 2 percent of the manganese were reduced and entered the metal, which averaged 94.2 percent iron and 1.0 percent manganese. At the end of the test the entire content of the crucible was poured, the metal being recovered as a button. A partial analysis of the slag was as follows: 5 percent iron, 29 percent manganese, and 31 percent silica. These tests show that manganese silicates are slowly reduced below 1,300° C.

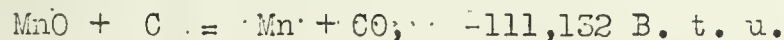
A series of reduction tests was made at higher temperatures on the manganese slag produced, as indicated above, from ferruginous manganese ores. The slag was crushed to minus 30-mesh, mixed with coke and limestone, and heated in hollow carbon electrodes. It was necessary to raise the temperature to about 1,450° C. before rapid reduction took place. Below 1,450° C. the reaction was so slow that little metal was produced.

Thermal Principles

The total heat absorbed by the two reactions,



is the same as the heat absorbed by the reaction,



That is, whether MnO is reduced by C or CO the reduction of each formula weight of manganese requires 111,132 B. t. u. or 2,020 B. t. u. per pound of manganese. Thus it may be seen that the reduction of MnO is very different thermally from the reduction of FeO, from which a pound of iron can be produced with a gain of 75 B. t. u. (If manganous oxide is reduced to manganese carbide, the reaction is slightly less endothermic; 1,920 B. t. u. per pound of manganese as the carbide).

Distribution of Heat in Pig-Iron
and Ferromanganese Furnace

A comparison between heat balances of pig-iron and ferromanganese furnaces shows why it requires less than one half as much coke to produce a ton of pig iron as it does to produce a ton of ferromanganese. Data in table 72 on the distribution of heat in pig-iron furnaces were taken from average values obtained by Royster, Kinney, and Joseph (324) for 14 American pig-iron furnaces. The heat balance for a ferromanganese furnace was calculated by Clements (49).

TABLE 72. - Distribution of Heat in Blast Furnaces Producing Ferromanganese and Pig IronHeat generated

B.t.u. generated per ton	Pig iron, ^{1/} 15,270,800	Ferro- manganese ^{2/} , 24,470,000
	Percent	Percent
Combustion of C to CO ₂	52.19	22.38
Combustion of C to CO	34.12	55.59
Heat in blast, including moisture	13.69	19.46
Heat generated by formation of silicates	-----	1.65
Heat generated by formation of carbides	-----	.92
Total	100.00	100.00

Heat consumed

Reduction of ferric oxide	42.92	3.57
Reduction of manganese oxides, phosphorus pentoxide, and silica	3.44	29.96
Dissociation of calcium carbonate	4.53	4.94
Dissociation of moisture	7.30	7.44
Carried off with metal	7.50	6.42
Carried off with slag	6.08	7.83
Carried off with dry gases	6.05	21.30
Carried off with moisture in top gas	4.15	2.97
Dissociation of manganese silicate	-----	.07
Dissociation of manganese hydroxide	-----	.82
Decomposition of calcium phosphate	-----	.11
Formation of calcium sulphide	-----	.81
Heat in flue dust	-----	.20
Heat absorbed by blast expansion	-----	.42
Heat in cooling water and radiation losses	18.03	13.14
Total	100.00	100.00

^{1/} Average data on 14 American blast furnaces reported by Royster, Joseph, and Kinney.

^{2/} Heat balance on a blast furnace by Clements.

Heat generated

Table 72 shows that 52.19 percent of the heat generated in the pig-iron furnaces was derived from combustion of C to CO₂, while in the ferro furnace only 22.38 percent of the heat was derived by combustion of C to CO₂. More than half the heat generated in the ferromanganese furnace was obtained by combustion of C to CO. The production of 1 ton of ferromanganese required 3,700 pounds of fixed carbon, from which 18,078,192 B. t. u. were generated, whereas the production of 1 ton of pig iron required approximately 1,700

pounds of fixed carbon, from which 13,179,300 B. t. u. were produced. These figures do not include the heat in the hot air used for combustion. In the ferromanganese furnaces, 4,885 B. t. u. were generated per pound of fixed carbon in the coke compared to a similar figure of 7,740 B. t. u. for the pig-iron furnaces. Thus it can be seen that about 1.6 times as much heat is generated per pound of carbon in the pig-iron furnaces. The production of a ton of ferromanganese on 3,700 pounds of fixed carbon is good practice, whereas the production of a ton of pig iron on 1,700 pounds of fixed carbon does not represent the best practice.

Johnson (180) states that maximum blast-furnace fuel economy depends (1) upon the development of the maximum possible amount of heat from each pound of fuel and (2) a certain irreducible proportion of the heat generated should be developed above the free-running temperature of the slag. The ferromanganese furnace violates both of these principles of fuel economy. The operation is characterized by high percentages of CO in the top gas because this gas is the end product of MnO reduction. The second principle of fuel efficiency enunciated by Johnson is based upon the desirability of maintaining hearth temperatures at which reactions that occur in this part of the furnace will go forward rapidly. In the pig-iron furnace the important hearth reactions are reduction of silica and desulphurization of the iron. High hearth temperature, which implies high temperatures of the tuyères, of the slag, and of the metal, would be very desirable in a ferromanganese furnace. In spite of the large amount of fuel used these temperatures in ferromanganese furnaces are low in comparison with those in pig-iron furnaces, as may be seen in table 73.

TABLE 73. - Hearth Temperatures

Furnace	Temperatures, ° C.		
	Tuyères	Slag	Metal
Foundry iron	1,748	1,553	1,493
Bessemer iron	1,733	1,513	1,466
Basic iron	1,669	1,522	1,468
Charcoal	1,669	1,451	1,415
Ferromanganese	1,550	1,426	1,386
Spiegeleisen	1,597	1,427	1,392

The average temperatures of the pig-iron furnaces are: Tuyères, 1,717° C.; slag, 1,529° C.; and metal, 1,476° C. Similar average values for ferromanganese furnaces are: 1,550°, 1,426°, and 1,386° C., respectively. Two factors are largely responsible for the lower temperatures in the ferromanganese furnaces. It has been pointed out that the reduction of MnO, whether by solid C or by CO, is highly endothermic. The absorption of heat in the lower part of the bosh, in the tuyere zone, and in the slag below the tuyères tends to lower what has been loosely called the hearth temperature. Slag, which is formed at low temperatures and which is fluid at temperatures just above its melting point, carries a comparatively small amount of heat into the crucible of the furnace. Less heat is absorbed above the fusion zone by the slag constituents. After fusion the slag passes rapidly over the coke lumps and absorbs a small amount

of heat in the bosh. Upon reaching the tuyère zone the large thermal head between the comparatively cold slag and the tuyères has a chilling effect upon the combustion zone. The net result is that the slag reaches the crucible at low temperatures. The temperature of the metal is governed largely by the temperatures of the tuyères and of the slag. This may be seen from the slag and metal temperatures reported in table 73. The slag has the predominating influence. Since the region of the furnace below the tuyères depends largely upon the slag and metal for its supply of heat, cold slag and cold metal mean comparatively cold temperatures in the crucible. Thus it may be seen that the low free-running temperature of the slag in ferromanganese and spiegel furnaces is not conducive to the high temperatures required for reduction of MnO . Since the temperature does not aid rapid reduction the alternative is to allow a longer time by bringing fresh slag from the bosh into the crucible at a slower rate.

Heat Consumed

Heat requirements for the calcination of carbonates and the dissociation of moisture are about the same in ferromanganese and pig-iron furnaces. The amount of heat carried off with the slag and metal is also about the same in the two types of furnaces. About 2.5 times as much heat is carried off with the exit gases from the ferromanganese furnace. This is the direct result of high top temperatures in the ferromanganese furnace and the larger volumes of exit gases per ton of metal produced. The volume of exit gases varies approximately as the fuel consumed per ton of metal, that is, from 2.5 to 5 times as much gas is produced in making 1 ton of ferromanganese. High top temperatures in ferromanganese furnaces are due to the small heat capacity of the charge. Furnas (103) found that the transfer of heat from gases to beds of broken solids depends upon the heat capacity of the bed per unit of volume. The specific heat of the ore and coke do not differ widely, both being on the order of 0.25. The weight of 1 cubic foot of stock in a pig-iron furnace is about 50 percent greater than in a ferromanganese furnace. This increase of 50 percent in heat capacity per unit volume of the charge permits more efficient acquisition of heat from the gas stream by the solids in pig-iron furnaces. Due to less efficient transfer of heat from the rising gas stream to the descending stock in the ferromanganese furnace the temperature of the top gas will range from 700° to 900° F. compared to 275° to 450° F. for pig-iron furnaces. Heat balances (table 72) for the two types of furnaces show that 3 to 4 times as much heat per ton of metal is carried off in the dry top gases issuing from the ferromanganese furnace.

Raw Materials for the Production of Ferromanganese

Ore

No rigid specifications can be set up for ores which may be used in producing ferromanganese because the value and feasibility of using a particular ore depend upon its cost and grade and the cost of other ores available at a specific point. The ore mixture usually consists of a number of ores, the proportion of each being regulated to meet rather definite chemical specifications for the over-all mixture. By blending a number of ores it is also possible to improve the physical character of the charge.

The proper ratio of manganese to iron is a foremost requirement of the ore burden. Standard ferromanganese averages 80 percent manganese and about 12.5 percent iron, a ratio of manganese to iron of 6.4 to 1. If no iron were obtained from the coke, a ratio of manganese to iron of 8 to 1 in the ore burden would be satisfactory if 80 percent of the manganese were recovered ($\frac{6.4}{0.8} = 8$). According to a careful chemical balance which Clements made from average data covering 2 weeks operation of a British furnace, about 25 percent of the total iron charged was derived from the coke (table 74). Only about 87 percent of the iron charged appeared in the alloy. The amount of iron derived from the coke will obviously vary with the fuel consumption and with the amount of iron in the coke ash. In general, the charge, including the coke, will contain 6 to 7 times as much manganese as iron. Allowing for a loss of 20 percent manganese and considering the iron obtained from the coke, the ore mixture should contain from 9 to 10 times as much manganese as iron.

Newton (274) has shown that the approximate grade of alloy can be calculated from the percentage of manganese and iron in the ore and from the recovery of manganese. Ferromanganese contains about 80 percent of manganese and 12 percent of iron. The percent manganese in the alloy may therefore be calculated as follows:

$$\text{Percent Mn in alloy} = \frac{(\text{percent Mn in ore} \times \text{recovery of Mn}) \times 0.92}{(\text{percent Mn in ore} \times \text{recovery of Mn}) + \text{percent Fe in ore}}$$

The application of the formula to an ore containing 49.5 percent Mn and 5.5 percent Fe, smelted with a recovery of 80 percent Mn, may be illustrated as follows:

$$\frac{(49.5 \times 0.80) \times 0.92}{(49.5 \times 0.80) + 5.5} = 81.0.$$

Thus it may be seen that according to this formula an ore mixture containing nine times as much manganese as iron will yield an alloy containing 81 percent manganese. If the amount of iron in the coke is increased 20 percent due to iron obtained from the coke, an alloy containing 79 percent manganese would be produced.

Silica Penalty

Ores low in silica are preferable for producing manganese alloys because the volume of slag increases as the silica in the charge increases. Increased slag means large losses of manganese, increased fuel, and loss of tonnage. From a study of data on furnaces operated during the War, Royster (322, p. 126) obtained a figure of 0.88 pound of carbon for each additional pound of slag. Newton assumed that 0.3 pound of carbon would be required to melt 1 pound of slag. The latter figure is a little higher than the theoretical values obtained by Evans and Bailey Scott, Deladriere, Lilot, Clements, and Johnson. These values have been averaged by Mott and Wheeler (264). These theoretical values are based upon the assumption that all the carbon charged is used in generating heat. This assumes that no carbon is involved in the reduction of manganese by endothermic reactions. The figure by Royster, based upon domestic practice with siliceous ores during 1917 and 1918, may be high for slag values ranging from 1,500 to 2,500 pounds but should be fairly accurate for higher slag volumes.

All important considerations, such as extra coke, extra flux, slower production, and higher costs above raw materials, are reflected in the price schedule (384) of ferromanganese ores which was adopted by the ferro-alloys committee of the American Iron and Steel Institute and was also approved by the War Industries Board in 1918. For each percent of silica in excess of 8 percent and up to and including 15 percent, the penalty was 50 cents per ton; between 15 and 20 percent silica, the penalty was 75 cents per ton. Langhorne (229) calculated the cost of producing 1 ton of ferromanganese from two ores containing 14.25 and 14.32 percent silica. The theoretical cost of producing 1 ton of ferromanganese from siliceous ore showed an increase of about \$10, assuming that a unit of manganese cost 60 cents in both cases. Since this additional cost would be distributed over about 2 tons of ore, it indicates a penalty of 50 cents per unit of silica in the ore.

Phosphorus

The amount of phosphorus in ferromanganese is governed almost entirely by the phosphorus content of the charge. Whether justifiable or not, the upper limit set by the trade for phosphorus in standard ferromanganese is 0.3 percent, which means that the ore mixture should average about one half of this or less. If the phosphorus in an ore exceeds 0.2 percent, there is considerably less freedom in the selection of other ores. The amount of phosphorus in the coke and flux must also be considered. A phosphorus penalty, adopted in 1918 by the ferromanganese committee of the American Iron and Steel Institute and approved by the War Industries Board, was as follows:

For each 0.01 percent in excess of 0.25 percent phosphorus there shall be a penalty against unit price paid for manganese of one-half cent per unit figured to fractions.

In view of existing conditions, and for the purpose of stimulating production of domestic manganese ores, there will be no penalty for phosphorus so long as the ore shipped can be used to advantage by the buyer. The buyer reserves the right to penalize excess phosphorus as above by giving 60 days' notice to the shipper.

The above prices to be net to the producer; any expense, such as salary or commission to buyers' agent, to be paid to the buyer.

Settlements to be based on analysis of ore sample dried at 212° F. The percentage of moisture in ore samples as taken to be deducted from the weight.

During the period 1911 to 1929 about 14 pounds of metallic manganese were used to produce 1 ton of steel. This is equivalent to 17.5 pounds of standard 80 percent ferromanganese. The addition of 17.5 pounds of ferromanganese, containing 0.3 percent phosphorus, to 1 ton of steel would increase the phosphorus by 0.0023 percent. Ingot samples often show a larger increase than this over samples taken before the addition of ferromanganese to the ladle because a small amount of phosphorus is reduced from the slag by carburizing agents. Herty (149) states that trebling the amount of phosphorus commonly found in ferromanganese and spiegel would not bother the steelmaker.

Physical Structure of Ores

The importance of physical structure is recognized in buying manganese ores. Hard, coarse ores are purchased for mixing with fine ores to curtail dust losses. A portion of the dust is recovered and charged in the raw form or briquetted. There is, however, an economic loss of manganese and an aggravation of troubles due to dust deposition in stoves and boilers.

The physical character of the ore is not important from the standpoint of resistance to gas flow, as the existing quantity of coarse coke results in a porous column offering little resistance to gas flow. Fine particles in the ore as charged or produced in the furnace make it difficult, however, to obtain uniform distribution of the gas, a matter very closely connected with the efficiency of the furnace as a heat interchanger between gases and solids. The physical character of the ore used in pig-iron furnaces places a limit upon tonnages; that is, a point is reached at which the pressure developed in the furnace interferes with the travel of the stock. This is not true of furnaces producing ferromanganese because recovery of manganese from the slag is the factor limiting the rate of blowing.

Coke

Low ash content is of foremost importance in selecting coke for use in furnaces producing ferromanganese because, in addition to higher fuel requirements and decreased tonnages, coke ash leads to an economic loss of manganese in the slag. With a consumption of 4,500 to 6,000 pounds of coke per ton of ferromanganese each percent increase in coke ash will result in an increase of about 100 pounds of slag. The harmful effects move in a vicious cycle. Increased ash lowers the fixed carbon in the coke, making more coke necessary, which, in turn, introduces more ash, requiring more coke.

It has been pointed out that in the range of 10 to 15 percent SiO_2 , each additional percent in the ore adds about \$1 to the cost of producing 1 ton of ferromanganese. Silica in the coke is more harmful than silica in the ore, because in addition to increasing the volume of slag it also replaces fixed carbon.

Coke with a low bulk density in the furnace is very desirable because this implies that the coke is strong yet porous and contains a limited amount of small sizes. Such coke, in addition to having thermal advantages, also has the advantage of producing a charge more permeable to gas flow.

Mott and Wheeler (264, p. 149) have summarized the most important properties of blast-furnace coke in the First Technical Report of the Midland Coke Research Committee, as follows:

From this survey of the more obvious chemical and physical properties of cokes it is evident that the most important are: Moisture and ash contents, hardness, and size. The value of a coke for blast-furnace use can be improved by decreasing the amounts of moisture and ash it contains, by increasing the hardness and by eliminating breeze.

In ferromanganese practice the attainment of high temperatures in the combustion zones is very important. In this connection these investigators state:

It is not possible at present to specify that property of a coke on which the attainment of high temperatures of combustion depends. Apparently the harder cokes yield the higher temperatures, whilst the size of the coke pieces is of importance.

The over-all rate of combustion of coke depends largely upon the rate at which air is supplied. Large sizes of less reactive coke that promote high concentrations of CO_2 by retarding the endothermic reaction between carbon and carbon dioxide aid in obtaining high temperatures.

In connection with the relation between coke hardness, specific gravity, and porosity, Mott and Wheeler (264) state:

From a study of the general properties of cokes we have tested, it appears that the average temperature of the fuel bed is lower the lower the impact hardness, real specific gravity, and total porosity.

This indicates that high porosity is an aid in obtaining high temperatures, provided it can be obtained without sacrificing strength or resistance to size degradation. According to Mott and Wheeler:

The porosity of coke appears to be of minor importance in blast-furnace practice. A high value seems to be desirable, but it is probably not worth while to attempt to increase the porosity directly.

The features of coke structure emphasized by these investigators are uniformity and paucity of enlarged pores and fractures. Although micro-structure most likely influences hardness and combustibility, no correlation of these properties was obtained.

Limestone

Silica in limestone decreases the amount of CaO and MgO available for fluxing the ore gangue and coke ash. The amount of limestone required and the slag volume increase sharply as the silica in the limestone increases. Although a poorer grade of limestone may be available at a lower price, it should not be used without the advice and consent of the furnace operator.

Operating Details of Ferromanganese Furnaces

The operation of furnaces producing ferromanganese is characterized by low blast pressure, regular settling of the stock, and loss of few tuyères. Little difficulty is experienced in controlling silicon. Sulphur presents no problem. Operating problems center upon ways and means of recovering in the metal the maximum percentage of manganese charged consistent with fuel consumption and output. Conditions that favor high recovery of manganese are: Small slag volume, a basic slag, high blast temperature, and coarse ores which do not break down excessively.

Percentage of Manganese in the Slag

The selection of raw materials is important in controlling slag losses. When the raw materials remain constant the loss of manganese in the slag is held to a minimum by varying the basicity of the slag, the rate of driving, and the hearth temperature. The optimum basicity of slag for maximum recovery of manganese will depend somewhat upon the amount of reduction of MnO above the tuyères and upon the temperature to which the blast can be heated. Brassert (37) states that through the use of the highest blast temperature (1,470° F.) the slag can be kept fluid enough in spite of being very "limey", and the manganese in the slag can be held uniformly below 6 percent. Royster (322, p. 129) found in studying data on ferromanganese furnaces operated during the war that when the average slag basicity was increased from 1.19 to 1.77 the manganese in the slag decreased from 14.2 to 7.4 percent. Data on British furnaces by Clements (49, p. 350) include slag basicities of 1.63, 1.91, 1.98, and 1.71, with corresponding manganese in the slag of 11.09, 7.27, 8.16, and 8.47 percent. As previously pointed out, low slag loss is usually accompanied by high volatilization loss.

Function of Bases in Reduction of Manganese

The addition of bases to charges of ferromanganese furnaces serves two purposes. It is well-known that the fusibility of a slag depends upon the character of the bases as well as upon the proportion of bases and silica. The total heat of solidification, which includes the heat of solidification plus the heat evolved in cooling from the melting point down to room temperature, is an index of fusibility. Hofman (164) has shown the influence of replacement of MnO by CaO upon the total heats of solidification or the fusibility of slags containing varying amounts of silica. In general, the total heat of solidification increases as MnO is replaced by CaO , thus showing that the presence of bases in the charge retards the formation of fluid

manganese silicates and hence aids in producing slag with higher free-running temperatures. Such slag reaches the crucible at a higher temperature which favors lower slag losses of manganese.

The slag in the crucible of a ferromanganese furnace is a solution of complex compounds. The activity of these compounds is affected by the amount of thermal dissociation. Moreover, manganese may exist as an infusible constituent. It appears that increased CaO makes the MnO more available for reduction. This may be accomplished by reaction with infusible compounds and the formation of slag constituents more susceptible to thermal dissociation.

Blast Temperature

About one half of the heat required to produce 1 ton of pig iron is introduced as sensible heat in the hot blast used to make 1 ton of ferromanganese. The quantity of heat thus introduced is not as important as the effect blast temperature has upon the prevailing temperature of the slag, metal, and tuyères. Richards (314) states that an increase of 100°C . in blast temperature raises the theoretical temperature before the tuyères 85°C . Royster, Joseph, and Kinney (326) found from observations on the tuyères of an experimental blast furnace that the temperature in front of the tuyères increased about 1° for every 2° rise in blast temperature. This observed change is about one half the theoretical calculated change. It is very difficult to isolate the effect of any one variable in blast-furnace practice. For this reason there is no established relation between blast temperature and temperature of the slag, metal, and tuyères. Reference has been made to the excellent practice reported by Brassert (37). With a blast temperature of $1,470^{\circ}\text{C}$., which is several hundred degrees higher than the average blast temperature, the coke per ton averaged 3,896 pounds for a month; the slag contained 5.81 percent manganese. A small amount of coal (288 pounds per ton) was used to reduce top temperatures and to minimize volatilization losses. With the use of a Cottrell treater much of the volatilization loss and dust loss may be recovered. The fact that the material so recovered is pyrophoric indicates that it is to a considerable extent metallic manganese.

Rate of Driving

Various methods have been used on rating the operating speed of blast furnaces, such as pounds of carbon per square foot of hearth area per minute, pounds of coke per cubic foot of working volume per 24 hours, and pounds of coke per square foot of bosh area per minute. Royster (322, p. 127) suggested the pounds of gross slag (MnO included in slag-forming materials) per square foot of hearth area per minute as a criterion for rate of driving. Gross slag includes the MnO equivalent of all of the manganese in the charge, which implies that all of the "metallic" manganese is produced in the hearth from the molten slag. This is a highly doubtful assumption. When this rule for speed of operation was applied to furnaces producing ferromanganese during the war, the manganese in the slag increased 1.34 percent for an increase of 1 pound of slag per minute per square foot of hearth area. The advantage of such a basis for governing the speed of operation is that it takes into account the size of the furnace as well as the grade of raw materials. Furnaces using more siliceous materials from which larger slag volumes would be produced would properly be blown at a slower rate.

TABLE 74. - Data on British Ferromanganese Practice (by Clements)

[illegible]

TABLE 74. - Data on British Ferromanganese Practice (by Clements)--Continued

Lines of furnace	I										II										III										IV																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																									
	Height of furnace	do	do	do	do	do	do	do	do	do	Height of furnace	do	do	do	do	do	do	do	do	do	Height of furnace	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do	do

Clements has compared the output of furnaces in terms of pounds of metal per square foot of hearth area per hour and per square foot of bosh area per hour (table 74) (49, table 37). On the basis of hearth area, the output of a group of American ferromanganese furnaces operated during the war ranged from 44 to 60 pounds compared with 110 to 136 pounds for four British furnaces. The metal produced per square foot of bosh area per hour ranged from 21 to 30 pounds in the American furnaces and from 28 to 34 pounds in the British furnaces. The slower rate of production in the American furnaces was due principally to the extensive use of domestic siliceous ores, whereas high-grade Indian, Caucasian, and Brazilian ores were used in the British furnaces. In general, a furnace capable of producing 500 tons of pig iron per day will have an output of 90 to 125 tons of ferromanganese. Recoveries can be increased by slower driving up to the point at which increased costs above raw materials make further recovery uneconomical.

Stack Loss

The amount of manganese carried out with the top gases either as ore dust or fine fumes, due to volatilization of manganese, usually is determined by difference; the manganese recovered in the slag and metal is deducted from the manganese charged. Values obtained in this way are subject to errors in calculating the weight of slag as well as to errors in the weights and analyses of raw materials and products. Exact determination of the amount of dust carried off in large volumes of gases is difficult, therefore dust losses are not susceptible to accurate measurement directly. The brown fumes issuing from the stove of ferromanganese furnaces and the necessity of frequently cleaning the stoves are evidence that substantial amounts of manganese are carried off with the gases in a finely divided condition. There is apparently no published information as to how much of the total stack loss consists of granular ore particles and how much consists of fine powder or fumes. Langhorne (229) states that minus 20-mesh particles do not seem to stay in the furnace, thus increasing the loss of manganese. He also points out that the physical character of the ore as charged does not always give an index of dusting characteristics because soft ores, which may be fairly coarse, are susceptible to size degradation in the furnace through contact with coke. The ratio of coke to ore by volume is much greater than in pig-iron furnaces and, hence, the opportunity for ore particles to be caught and crushed between lumps of coke is much greater. Joseph and Barrett (186) found that five imported manganese ores were little affected by cold mechanical work, but the average particle size was decreased 40 to 50 percent as a result of a moderate amount of mechanical work at 1,400° F. Nearly 50 percent of one ore was finer than 14-mesh after a decrepitation test. These tests indicate that manganese ores may break down a great deal when heated and subjected to the abrasive action of coke in a ferromanganese blast furnace. It seems likely that a substantial portion of the so-called stack loss is an ordinary dust loss, which varies depending upon the amount of fine material in the ore burden and upon the extent to which the ore mixture breaks down in the furnace.

The volatilization of manganese should increase as the hearth temperature increases. Royster found no systematic variation of stack loss with increased hearth temperature, as indicated by the high-silicon metal, excess carbon, basic slag, and low manganese in the slag. This apparent lack of relationship

is not surprising, considering the chances for error in arriving at stack losses by difference. It is known from practice that dust losses vary widely with the character of the ore and the distribution of stock which governs channeling of the gases. The ores used in the furnaces investigated by Royster came from many sources and varied widely in character. Losses due to dusting of the ore might very easily mask variations in volatilization losses. Although the exact amount of manganese carried off with the gases is a quantity not susceptible to accurate measurement, it is an important item in furnace practice, ranging from 5 to 20 percent of the manganese charged.

Composition of Dust from Ferromanganese Furnaces

The composition of the dust recovered from ferromanganese furnaces raises some interesting questions. An examination of the analyses of raw materials and of the dust samples given in table 74 shows that the dust contains more iron in proportion to manganese than the charge. Why should iron, a less-volatile metal occurring in much smaller amounts in the charge, be concentrated in the dust? The ratio of manganese to iron in the charge of three British furnaces ranged from 8.92 to 9.21, exclusive of coke, and from 6.6 to 6.8, inclusive of coke. The reported composition of dust samples taken from these same furnaces shows a manganese-iron ratio of 4.1:5.9. One would naturally expect to find a concentration of manganese in the dust rather than a concentration of iron. Royster found that 6.5 percent of the iron charged into a furnace did not appear in the slag and metal. It appears that substantial amounts of coke ash, formed on the surface of the coke as a fine powder, are carried by the gases up through the porous and comparatively hot column of stock and out of the furnace. It is generally considered that the ferro-grade ores should contain about nine times as much manganese as iron. If the iron in the coke is taken into consideration, the average charge contains 6 to 7 times as much manganese as iron. The evidence points very strongly to a loss of at least a portion of the iron in the coke in the manner indicated.

Exclusive of the coke, the charge of two British furnaces operating on high-grade ores contained about five times as much manganese as silica (table 74). Dust samples from these same furnaces contained 1.5 to 1.7 times as much manganese as silica. There is a concentration of silica in the dust which cannot be explained as a dust loss from the charge, unless an air classification of silica is assumed to occur. This concentration of silica in the dust is more likely due to coke ash carried off in the exit gases. Alumina is also concentrated in the dust in a similar way.

Limestone is not readily abraded by coke before calcination, but after calcination is susceptible to abrasion. The presence of some 12 to 15 percent calcium oxide, compared to 20 to 25 percent MnO in the flue dust from furnaces producing ferromanganese, is due to abrasion followed by dusting.

The relative amount of Fe_2O_3 , SiO_2 , CaO , Al_2O_3 , and MnO in the dust samples, reported by Clements, suggests that the loss of manganese by volatilization has been given undue emphasis in the literature dealing with the production of ferromanganese in the blast furnace. Resistance of manganese ores to size degradation in the furnace and the size of the ore as charged have an important bearing on the amount of dust produced and suggest more careful size preparation of the ores as a means of decreasing the so-called "stack losses."

It appears that much of the so-called volatilization loss in the form of fine fumes may be due to oxidation before the tuyeres, followed by dusting.

Furnace Lines

Furnaces producing ferromanganese operate smoothly without hanging or slipping, a natural result of a porous charge which offers little resistance to gas flow and results in low pressures. The slow rate at which air is supplied also tends to produce low pressures, which promote regular settling of the charge. As a result of this freedom from mechanical troubles, the shape of furnaces producing ferromanganese has not received the same consideration as the shape or lines of pig-iron furnaces. It is common practice in plants which produce pig iron and ferro-alloys to put furnaces on ferromanganese or spiegel after the lining has been worn to such an extent that smooth operation on pig iron is impossible. Smaller furnaces, in general, are used in producing ferro-alloys.

Location of Plants Producing Ferromanganese

Smith (352) reports that ferromanganese was produced at the following domestic plants in 1929:

Bethlehem Steel Co., Johnstown, Pa.
 Colorado Fuel & Iron Co., Pueblo, Colo.
 Lavino Furnace Co., Reusens, Va.
 Ohio Ferro-Alloys Corporation, Philo, Ohio.
 Pittsburgh Metallurgical Co., Niagara Falls, N. Y.
 Tennessee Coal, Iron & Railroad Co., Bessemer, Ala.
 United States Steel Corporation, Etna and Pittsburgh, Pa.

Electric furnaces were used by the Ohio and New York companies. In 1928, ferromanganese was produced at seven furnaces by three companies.

Production of Ferromanganese in Electric Furnace

Ferromanganese can be produced in the electric furnace from a proper charge of ore, coal or coke, and limestone. If arcing from the electrodes to the bath is avoided volatilization losses are not excessive. As in other electric-furnace operations, cheap power is necessary for economical operation. The important features of the operation are: Power and electrode requirements, size and type of furnace, amount and type of reducing agent, and character of slag losses.

Keeney (196) states that the power per ton of ferromanganese ranges from 4,000 kw.-h. in a 3,000-kw. furnace to 7,000 kw.-h. in a 1,000-kw. furnace. The electrode consumption is 150 to 250 pounds of amorphous carbon electrodes per ton. Power and electrode requirements will vary with the grade of raw materials used. Gillett and Williams (112) used 431 pounds of graphite electrodes per ton per 1,000 kw. of power in calculating the cost to produce ferromanganese from siliceous domestic ores.

Keeney and Lonergan (197) have described a 1,200-kva furnace and an 1,800-kva furnace in producing ferromanganese. The description includes details of the two rectangular, three-electrode furnaces, transformers, electrode holders, and secondary bus systems. Reducing agents, calculations of the charge, and method of operation are discussed. Difficulties of operation which may be encountered include formation of carbides, "blowing" failure of bottom, tapping, failure of electrode holders, and bridging of the charge.

Voltages (electrodes to the charge) range from 55 to 100, depending upon the size of the furnace. Charging may be continuous or intermittent. Coke, coal, lignite, and charcoal may be used as reducing agents. The quantity required varies, depending upon the grade of raw materials. Magnesite or carbon may be used to make the furnace bottom. Metal and slag are usually tapped at 2-hour intervals.

Keeney (196) reports an average slag containing 35.4 percent CaO, 5.81 percent MgO, and 27.79 percent SiO₂, or a ratio of $\frac{\text{CaO} + \text{MgO}}{\text{SiO}_2}$ of 1.48:1.

Slags somewhat more basic than this are preferred in the blast furnace. A basic slag will lead to more complete elimination of manganese from the slag, provided ample time is allowed. If the rate of charging is too rapid additional bases may lower the recovery, because a slight decrease of manganese in the slag would be offset by the increase in the weight of the slag. With a proper adjustment, however, of tonnage for a given furnace, basic slags aid in controlling silicon in the alloy and decrease the loss of manganese in the slag.

Manganese ores break down to dust even in the electric furnace. It is not possible therefore to differentiate between dust losses and losses by volatilization. The combined loss from dusting and volatilization is 5 to 10 percent. Slag loss depends to a large extent upon the silica in the ore, which, in turn, governs the weight of slag per ton of alloy. The total losses will be 10 to 30 percent depending upon the grade of ore used. With high-grade ores a 90 percent recovery of manganese can be realized. When smelting siliceous ores the loss due to volatilization and dusting will amount to one-fourth to one-sixth of the total conversion loss, whereas with the best grades of ore the loss of slag may about equal the combined loss by dusting and volatilization.

Production of Low-Carbon Ferromanganese

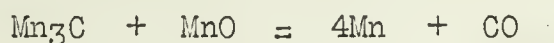
Standard ferromanganese, containing about 7 percent carbon, cannot be used in producing extremely low-carbon steel. Either medium- or low-carbon ferromanganese is used in making additions to such steel. Very little recent information has been published upon the production of low-carbon ferromanganese.

Roberts and Wraight (316, pp. 250-282) investigated various methods for decarburizing ferromanganese, which may be grouped as follows:

1. Replacement of combined carbon by silicon.
2. Replacement of combined carbon by aluminum.
3. Cementation in various metallic oxides.
4. Fusion at high temperatures with lime.
5. Bessemerization of the alloy with air, CO₂, and H₂.
6. Oxidation of carbide by manganous oxide.

The most satisfactory results were obtained by the sixth method, that is, fusion with manganous oxide.

As pointed out by Coutagne (60), Moissan showed by obtaining alloys with 3 to 4 percent carbon by reduction of MnO with carbon in the presence of excess MnO that the reaction



will proceed at the high temperature attainable in the electric furnace. He also states that if manganese carbide is formed first, which might be the case, it is subsequently partly refined, or, if manganese is formed, first, carbon absorption will depend upon the amount of oxide present. Whatever the mechanism of reduction may be, this is an expensive method for producing manganese with low carbon content.

The refining action of manganous oxide upon manganese carbide is the basic feature of patents by Humbert (173) and Hadfield (134). Humbert's patent proposes to decarburize commercial ferromanganese by electrically heating it to temperatures above 1,700° C. in the presence of manganous oxide. Small coke or ferrosilicon is used to reduce part of the manganous oxide. According to Hadfield's procedure, commercial ferromanganese is treated with manganese ore and enough carbon to reduce the higher oxides in the ore to manganous oxide. After gradually building up a bath of ferromanganese and slag rich in MnO and therefore viscous, ferrosilicon is added to reduce part of the manganous oxide. The silica formed by reduction of MnO with silicon increases the fluidity of the slag. Lime additions are made to control the fluidity of the slag. Calcium silicide may be added shortly after the final addition of ferrosilicon. The materials used in a heat, cited to illustrate the process, were as follows: 15 cwt. of ferromanganese, 5-1/2 cwt. of manganese ore, 44 pounds of 50 percent ferrosilicon, and 17 pounds of calcium silicide. The alloy produced contained 71.6 percent manganese and 1.54 percent carbon.

Coutagne (60) states that the most likely process and the one most widely used consists in refining silicomanganese by treatment with manganese ore. The silicomanganese is first produced in the electric furnace. A charge of such an alloy is next treated with manganese ore, the materials involved in heat cited being as follows:

	Lb.
Silicomanganese (2.7 percent C, 60 percent Mn, and 13 percent Si	1,320
Turnings	110
Mineral (35 percent Mn)	880
Lime	880

The metal produced from such a charge weighed 1,540 pounds and analyzed as follows: 2.5 percent carbon, 53 percent manganese, and 1.77 percent silicon. If an alloy containing 75 percent manganese, 20 to 25 percent silicon, and 1.0 percent carbon is used, this mode of operation is reported to yield alloys containing less than 1.0 percent carbon.

Becket (19) has recently patented a similar process for producing low-carbon ferromanganese. In two examples which he cited, the following materials were used:

	<u>No. 1</u>	<u>No. 2</u>		<u>No. 1</u>	<u>No. 2</u>
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Weight of alloy .. lb.	14,948	16,443	Weight of alloy ... lb.	26,935	26,389
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Analysis of charge, percent:

<u>Alloy:</u>			<u>Ore:</u>	
Mn	64.44	68.30	56.50	(largely as MnO ₂)
Si	24.66	23.13	2.17	Fe
C84	.67		

Balance largely iron.

Analysis of metal produced, percent:

Mn	80.11	83.91
Si	1.22	1.83
C	1.25	1.08

Balance largely iron.

The manganese-silicon alloy and the manganese ore were ground to pass 100 mesh and thoroughly mixed and dried. The silicothermic reduction of the manganous oxides was self-propagating after the charge was locally heated to a temperature sufficient to initiate the reaction. No apparatus is required except a refractory pot or vessel to hold the charge. The carbon content of the manganese alloy is controlled by selecting a manganese-silicon alloy of appropriate composition. For example, Beckett states that a manganese-silicon alloy containing 1.0 percent carbon and 20 percent silicon may be used in producing an alloy with about 1.5 percent carbon, while a manganese-silicon alloy containing 0.7 percent carbon and 24 percent silicon yields a product containing about 1.0 percent carbon. A product still lower in carbon is produced by using a manganese-silicon alloy containing more silicon and less carbon.

The slag produced in the heats cited by Beckett contained about 65 percent MnO. It can be smelted in an electric furnace to produce manganese-silicon alloys similar in composition those described. Slag from this operation contains about 15 percent MnO and is discarded.

The production of low-carbon manganese alloys on a commercial scale is a comparatively new development. Few data have been published regarding the practical details of the operation.

PRODUCTION OF SPIEGELEISEN

The factors that characterize good practice in producing ferromanganese, the 80 percent alloy, apply in a modified degree to the production of spiegeleisen, the 20 percent alloy. Low-silica charges, slow blowing, and regulation of the slag basicity to minimize slag losses are important in the production of both alloys. The fuel requirements, tonnage, and recovery depend upon the amount of silica in the charge. Royster (322, p. 117) reported an average fuel consumption of 3,950 pounds of coke per ton of 16 to 20 percent alloy for furnaces operating during the World War. The slag averaged 9.68 percent manganese. This fuel consumption and the percentage of manganese in the slag are much higher than was obtained in furnace practice summarized by Brassert (37), who reported an average figure of 2,858 pounds of coke per ton of 19 percent alloy and 2.18 percent manganese in the slag. High prices during the war stimulated efforts to produce tonnage often at the sacrifice of manganese recovery. This condition, together with the siliceous charge used at that time, accounts for the poorer practice reported by Royster. According to Brassert, a furnace having a normal daily capacity of about 500 tons of pig iron will produce about 275 tons of spiegel per day. This, however, implies the use of good raw materials and stoves which will produce high blast temperatures.

Production of High-Silicon Spiegel

The work of Herty and Fitterer on the deoxidation of steel with manganese-silicon alloys has created interest in high-silicon spiegel. They have reported (152) that a ratio of manganese to silicon of 4.5:6.0 gave the best results in deoxidizing both basic open-hearth and acid open-hearth heats ranging from 0.12 to 0.6 percent carbon. Such a ratio of manganese to silicon would require about 4 percent silicon in normal spiegel containing 20 percent manganese.

Restrictions as to grade of manganese alloys were modified during the World War to permit wider use of lean and siliceous domestic ores. The standard for spiegel was lowered from 20 to 16 percent manganese. The production of occasional lots of high-silicon spiegel of this grade was a natural consequence of a furnace practice evolved rather quickly for treating siliceous ores. Operating data on the production of such lots of spiegel are given in table 75. The ratios of manganese to silicon are 4.8, 3.4, 5.5, and 3.7 for test periods 41, 42, 68, and 70, respectively. No mention is made of any peculiarities or operating difficulties encountered in these test periods. One may assume, therefore, that the operation was similar to normal spiegel practice. The manganese content of the metal could have been raised to 20 percent by use of an ore mixture containing a higher ratio of manganese to iron. A higher fuel consumption and lower rate of driving would have increased the poor recovery of manganese in furnace C.

An examination of the data in table 75 indicates that temperature and rate of driving have foremost importance in producing high-silicon spiegel. The two

test periods on furnace C show the importance of temperature. About 500 pounds of additional carbon and somewhat higher blast temperature increased the silicon in the metal from 3.1 percent for test period 41 to 4.9 percent for test period 42. This increase of 1.8 percent silicon in the alloy produced from the same raw materials in the same furnace illustrates the importance of temperature in the reduction of SiO_2 . The decrease in output from 44.7 tons per day in period 41 to 30.2 tons in period 42 indicates that the production of high-silicon spiegel for a given furnace and set of raw materials will be less than that of standard spiegel. Further evidence of the decrease in tonnage as the silicon in the alloy increases appears in test periods 68 and 70 of furnace R, which produced 71.4 tons per day of spiegel containing 2.9 percent silicon and 59.3 tons of spiegel containing 4.4 percent silicon.

TABLE 75. - Operating Data from Blast Furnaces
Producing High-Silicon Spiegel
 (by Royster) 1/

Period no. Furnace	41 C	42 C	68 R	70 R
1. Ore pounds per ton	5,192	5,795	5,495	5,355
2. Coke do	3,327	3,831	5,853	5,048
2a. Coal do	749	926	-----	-----
3. Limestone do	1,568	2,052	3,728	3,480
Ore analysis:				
4. Mn percent	12.72	12.77	10.25	9.06
5. Fe do	36.72	36.30	32.55	31.25
6. SiO_2 do	17.46	16.65	19.97	16.48
7. Spiegeleisen tons per day	44.7	30.2	71.4	59.3
8. Slag pounds per day	2,614	2,990	4,585	4,735
9. Mn in alloy percent	14.9	16.5	15.9	16.1
10. Si in alloy do	3.1	4.9	2.9	4.4
11. Blast temperature °F.	1,234	1,304	768	909
12. Top temperature . do	613	749	535	483
13. Air per minute .. cubic feet	8,660	8,740	28,800	21,300
Slag analysis:				
14. CaO percent	26.0	24.3)	44.0	46.1
15. MgO do	10.1	11.1)		
16. Al_2O_3 do	10.3	13.3	9.7	11.3
17. SiO_2 do	38.3	37.2	36.5	36.3
18. Mn do	10.79	10.07	6.38	3.66
19. Carbon pounds	3,292	3,740	4,790	4,110
20. Mn charged do	660	740	563	486
21. Mn to alloy percent	49.0	50.6	63.4	73.6
22. Mn to slag do	42.6	40.7	51.8	35.6
23. Mn lost in stack. do	8.4	8.7	-15.2	- 9.2

1/ Royster, P. H., Production of Manganese Alloys in the Blast Furnace: Chap. 8, Bull. 173 (Manganese--Uses, Preparation, Mining Costs, and the Production of Ferro-Alloys), Bureau of Mines, 1920, table 18, pp. 116-117.

The recovery of manganese in furnace R was much higher than in furnace C, --63.4 and 73.6 percent compared to 49.0 and 50.6 percent. This is attributed to the use of more fuel per ton of metal in furnace R.

Basicity of Slag in Producing High-Silicon Spiegel

Slag basicity is important in controlling the silicon in all metal produced in the blast furnace, whether it be pig iron or alloys of manganese. The reduction of silica is not, however, controlled entirely by the character of the slag but by proper adjustment of temperature and rate of blowing, as well as slag basicity. For example, the ratios of bases to acids, $\frac{\text{CaO} + \text{MgO}}{\text{SiO}_2}$, was about 1.2 for test periods 41 and 42. Due, however, to an increase in the fuel, the silicon in the metal was 1.3 percent higher in period 42 (see table 75).

The slag basicity averaged 1.26, and the percent of silicon in the metal averaged 1.77 for a group of furnaces producing 16 to 20 percent spiegel during the World War. Two of these furnaces produced metal averaging 3.8 percent silicon, with an average slag basicity of 1.22.

From the meager operating data available it appears that spiegel containing 20 percent manganese and 4 percent silicon can be produced in the blast furnace from domestic ores without any unusual operating difficulties. Since manganese aids carburization and silicon retards it, one might expect that unusual amounts of "fish" or precipitated carbon would be formed in producing high-silicon spiegel. This assumes that carburization precedes the reduction of silica. The introduction of 4 percent silicon into metal containing 15 to 20 percent manganese and 5 to 6 percent carbon would decrease the carbon to about 3 percent.

Carbon precipitated as a result of increased silicon content or of a drop in temperature as the metal moves from the combustion zones to the crucible could be consumed by unreduced oxides in the slag or by oxygen from the blast.

During the blow-in period of the experimental blast furnace for investigating the smelting characteristics of high-manganese slag (see p. 230), several casts of high-silicon spiegel were produced over a 10-hour period. The average percentage composition of the metal and slag follows:

Metal, percent			Slag, percent				
Mn	Si	C	Mn	SiO ₂	Al ₂ O ₃	CaO	MgO
20.2	3.7	3.2	9.7	3.0	23.0	31.2	3.9

The ratio of bases to acids, $\frac{\text{CaO} + \text{MgO}}{\text{SiO}_2}$, for the average slag is 1.17 or very close to the basicity of the slag reported in table 73. No unusual features of operation were observed during the production of high-silicon spiegel in the experimental blast furnace.

As pointed out by Coutagne (60, p. 212), the production of silicon-spiegel (10 to 12 percent Si and 18 to 20 percent Mn) in the blast furnace presents difficulties because the recovery of manganese requires a basic slag, while the reduction of silica is favored by an acid slag. This condition requires extra heat, which increases the cost of upkeep on the furnace lining and the loss of manganese by volatilization. As the ratio of silicon to manganese increases the observations noted above can be applied more rigorously. They apply to some extent to the production of spiegel containing 20 percent manganese and 4 percent silicon but are not important enough to throw the manufacture of this alloy from the category of the blast furnace into the category of the electric furnace.

Coutagne (60, p. 390) states that silicospiegel (10 to 12 percent Si) was made in the blast furnace by Pourcell in 1875 but the electric furnace offers advantages in cost and ease of producing a wide variety of alloys. Power requirements range from 2,500 to 3,000 kw.-h. when smelting a charge of high-grade manganese ore, turnings, quartz, and coke. The recovery of manganese from such a charge will range from 80 to 85 percent. If the charge consists of siliceous ore, turnings, limestone, coke, and spar the recovery is about 10 percent less.

Use of Ferruginous Manganese Ores in Production of Spiegel

The black ores of the Cuyuna range in Minnesota contain about three times as much iron as manganese, the approximate ratio necessary in producing 20 percent spiegel. These ores were used extensively during the World War but have not been mined extensively since that time because of their high silica content and comparatively high mining costs. The removal of silica from these ores will provide a supply of concentrate from which spiegel and high-silicon spiegel can be produced. The relative cost of units of manganese in the form of ferromanganese and in the form of spiegel required for the deoxidation of steel has no doubt been a factor in the transportation of spiegel over long distances. It appears therefore that the production of spiegel in middle western steel plants would stimulate wider use of this alloy.

A mill for concentrating the black ores of the Cuyuna range district was operated in 1931. The flow sheet is the result of work (67) at the Mississippi Valley Experiment Station of the United States Bureau of Mines in cooperation with a Minnesota mining company. Ore crushed to 5 mesh is classified and then tabled; the slimes from the tables are treated by flotation for additional recovery of fine iron and manganese. The daily capacity is about 500 tons of concentrate containing approximately 39 percent iron, 18 percent manganese, and 10 percent silica. Such a concentrate is well suited for the production of 20 percent spiegel. The ratio of manganese to iron would permit the production of a somewhat richer spiegel if this were desired.

Concentrates from the black Cuyuna ores could also be used in producing high-silicon spiegel (4 to 5 percent silicon). Concentrates would be superior to the crude ore for this purpose. Temperature is very important in producing high-silicon alloys. The required temperature can be obtained with less fuel when there is less silica in proportion to the iron and manganese in the charge. Lower silica would also permit higher recoveries and larger tonnages.

Reduction of silica is a function of the ratio of silica in the charge to the bases and is not determined by the total amount of silica present.

There are deposits of ferruginous manganese ores in other parts of the country which could be used in producing spiegel. Transportation charges to steel centers would be prohibitive in many cases.

PRODUCTION OF HIGH-MANGANESE BASIC PIG IRON

Manganiferous iron ore shipped from the Lake Superior district in 1929 contained 157,590 tons of metallic manganese, about 45 percent of the total tonnage of manganese used that year in the form of ferro and spiegel. The metallic manganese derived from manganiferous iron ores increased more than tenfold from 1911 to 1929. Manganiferous iron ores are mixed with iron ores for the production of high-manganese basic pig iron (1.5 to 2.25 percent manganese). A tenfold increase in the quantity of manganese contained in the pig iron used in producing basic open-hearth steel did not decrease the amount of manganese added in the form of ferromanganese and spiegel. For example, in 1911 shipments of manganiferous iron ores from the Lake Superior district (largely from the Cuyuna range, Minnesota) contained 14,629 tons of metallic manganese. An average of 14.1 pounds of manganese was added as ferromanganese and spiegel to produce 1 ton of steel that year. Although shipments of manganiferous iron ore in 1929 contained 157,590 tons of metallic manganese, 14 pounds of manganese were used in the form of spiegel and ferromanganese to produce 1 ton of steel. It appears that increases in manganese specifications in steel have been sufficient to offset the small amount of residual manganese derived from basic pig iron.

Weld and others (385) concluded that the increased use of high-manganese pig iron probably would replace 15 percent of the manganese needed as ferromanganese and spiegel. At the time of their survey of the domestic manganese situation the trend in manganese consumption appeared to be downward. In view of this trend the committee adopted 13 pounds as the probable average future consumption of manganese per ton of steel. Information now available indicates that this figure should be about 14 pounds. The evidence points strongly to the fact that only a very small amount of the manganese derived from manganiferous iron ores reaches the steel ingot.

Recovery of Manganese in Basic Pig-Iron Practice

About 75 percent of the manganese in the charge is recovered in the metal in basic pig-iron practice. The amount of retained residual manganese in basic open-hearth steel depends very largely upon the extent to which plant practice favors a high percentage of MnO and a low percentage of FeO in slag at the end of the heat. The percentage of manganese in the slag depends upon the silicon in the metal, the silica content of the iron ore, any other factors that influence the slag volume, the percentage of manganese in the metal, and the proportion of iron and scrap. The FeO in the slag depends upon the amount of ore added and upon the slag volume. Manganese is eliminated during the early part of the heat; therefore, if a part of the slag is run off before the heat is finished the percentage of MnO in the finishing slag and the residual manganese will be low. Kinney (204) has published a chemical balance of an open-hearth heat made under conditions which favored high residual manganese. The steel at tapping contained 0.4 percent manganese,

which represented 27.6 percent of the manganese charged. Pazetti (297) reports a month's practice using pig iron containing 1.65 percent manganese and no run-off slag. The residual manganese averaged 0.22 percent. He states further:

In a plant using run-off slag, the record of five carefully observed heats showed average residual manganese to be 0.09 percent when the average manganese in the pig iron was 1.69 percent.

It is obvious that under some conditions virtually all the 150,000 tons of manganese derived in recent years from manganiferous iron ore found a resting place either in blast-furnace slag heaps or in those obtained from open-hearth furnaces. Hoyt (169), in discussing the effect of the use of manganiferous iron ores on manganese requirements, states:

Present data indicate, unfortunately, that no material decreases in the amount of manganese required and no material increase in the recovery of manganese in the additions may be expected, so that the advantages are derived not from a decreased consumption but from the form in which it can be added.

Data on the increased quantity of manganese derived from manganiferous iron ores and the small changes in the manganese added as ferromanganese and spiegel from 1911 to 1929 confirm this viewpoint. Commercial expediency determines the practice which will prevail in individual plants. It appears that, in general, benefits due to secondary causes, such as slag fluidity and sulphur elimination, rather than those derived from residual manganese have been responsible for the use of high-manganese basic pig iron.

Desirability of Manganese in Basic Pig-Iron Practice

Sulphur control belongs essentially to the blast furnace. It can be eliminated in the open-hearth furnace by the use of additional limestone or ferromanganese, but both methods are costly and uncertain. Manganese in the charge aids in eliminating sulphur in the blast furnace. Herty and Gaines (153, 154) have shown the underlying cause for further elimination of sulphur by manganese during the time the metal is in the transfer ladles between the blast furnace and the open-hearth mixer. Wheaton (391) investigated the decrease in sulphur during the transfer of high-manganese basic pig iron from the blast furnace to the open-hearth mixer. His results follow:

	Manganese blast furnace, percent	Sulphur at blast furnace, percent	Sulphur at open-hearth mixer, percent
Average, 29 tests,	1.39	0.0867	0.041
Average, 89 tests,	1.72	.070	.035
Average, 26 tests,	2.13	.073	.031

Smith (350) states that high-manganese pig iron has the following advantages in the blast furnace: Increased rate of driving; high bottoms reduced; less runner and ladle scrap; higher blast temperatures possible;

lower fuel consumption; less flue dust; and elimination of sulphur. He also states that irregular content of manganese in the ore (including iron ore) shipped is a disadvantage due to irregular content of manganese in the pig iron. Excessive percentages of manganese may also lead to breakouts due to low bottoms and corrosion of brickwork.

In open-hearth practice, advantages and disadvantages were given as follows: Advantages--sulphur elimination, improved quality of steel, ferromanganese saving, less lime and fluorspar, and increased tonnage; disadvantages--higher metallic losses and increased lining corrosion.

It has been rather generally assumed that residual manganese protects the steel against oxidation. Herty (150) states:

Residual manganese does not serve as a protector against oxidation of the metal. Since the concentration of MnO in the metal depends on the concentration of MnO in the slag, it is possible to have the same concentration of FeO in the metal with 0.1 percent manganese as with 0.4 percent manganese, provided only that the concentration of MnO in the slag is four times as high in the second case as in the first. High residual manganese may denote a bath low in iron oxide, but the low iron oxide is the agent and not the result of high residual manganese.

The advantages of high-manganese, basic open-hearth charges, as given by Herty, are: (1) Increased fluidity of the slag, enabling a more basic slag to be carried, and increased speed of working; (2) increased ratio of MnS to FeS in the metal, giving the final addition of manganese a better chance to eliminate sulphur; and (3) decreased additions of ferromanganese.

Producers of manganiferous iron ores have noticed a decrease in the demand for this type of ore. Records, compiled by the Lake Superior Iron Ore Association, show that in 1927 the tonnage of manganiferous iron ores amounted to 4.6 percent of the total ore shipments from the Lake Superior district. In 1930, this figure was 3.6 percent, a decrease of about 22 percent. Some plants have found that alkali additions to the ladles give them the necessary control over sulphur. More extensive use of basic open-hearth slag has also lessened the demand for manganiferous iron ores.

THREE-STEP PROCESS FOR PRODUCING FERROMANGANESE FROM MANGANIFEROUS IRON ORES

Acknowledgments

The writer wishes to acknowledge the valuable assistance of E. P. Barrett, C. E. Wood, and F. W. Holbrook in connection with the experimental work hereafter reported in this chapter.

Due to their size, accessibility, and state of development the manganiferous iron ore deposits of Minnesota have attracted attention as a potential source of manganese. Zapffe (405) states:

The Cuyuna district has a reserve tonnage of ore of 44,000,000 tons. If it averages 9 percent in manganese as the ore produced at present, it has a reserve of 3,960,000 units (tons of metallic manganese).

The United States Bureau of Mines has investigated the feasibility of producing ferromanganese from manganiferous iron ores containing 6 to 10 percent manganese, 40 to 45 percent iron, and 0.2 to 0.3 percent phosphorus. The ore was first smelted in an experimental blast furnace from which high-phosphorus spiegel (12 to 15 percent manganese and 0.5 percent phosphorus) was obtained. In the second step the spiegel from step 1 was treated in a small basic-lined converter, small open-hearth furnaces, and small electric furnace to separate the manganese from the phosphorus and the iron. The object was to obtain the manganese in the form of slag containing manganese, iron, and phosphorus in such proportions that the slag could be used as an artificial manganese ore. In the third step the high-manganese slag from step 2 was charged into the experimental blast furnace and converted into ferromanganese. Photographs of the furnaces used, arranged as a flow sheet, appear in figure 28. The small converter is not shown with the open-hearth and electric furnaces used in the second step because it proved unsatisfactory for separating manganese from the phosphorus and the iron.

Blast-Furnace Tests

In present practice, brown manganiferous iron ores are mixed with iron ores to produce basic pig iron. There was some question whether the manganiferous iron ores could be smelted by themselves due to high alumina in proportion to silica and high moisture content. Inasmuch as metal was needed for investigating subsequent steps of the process, it was decided to produce the metal from the ore rather than make it up synthetically from spiegeleisen ferrophosphorus, and pig iron. The cost of smelting these ores on an industrial furnace scale was prohibitive; therefore an experimental blast furnace was developed partly for this problem and partly for investigating problems fundamental to the production of pig iron.

A detailed description of the experimental blast furnace and the results of the test have been published (187). The furnace was operated continuously for about 1 month without any difficulties. Over 300 tons of ore were put through the furnace in producing about 130 tons of metal containing 12 to 15 percent manganese and 0.55 percent phosphorus. Rather than attempt to select an average or typical ore, material unusually high in alumina and moisture was chosen. Many Cuyuna ores which would produce slags in the normal range of 10 to 15 percent alumina present no smelting problem. Slag containing from 9.4 to 21 percent alumina was produced. A study of 278 slag samples showed that, as a whole, the effect of alumina was not very pronounced. If the alumina content of the slag was low, manganous oxide in the slag fell off more sharply as the basicity ($\frac{\text{CaO}}{\text{SiO}_2}$) increased than when the alumina was high.

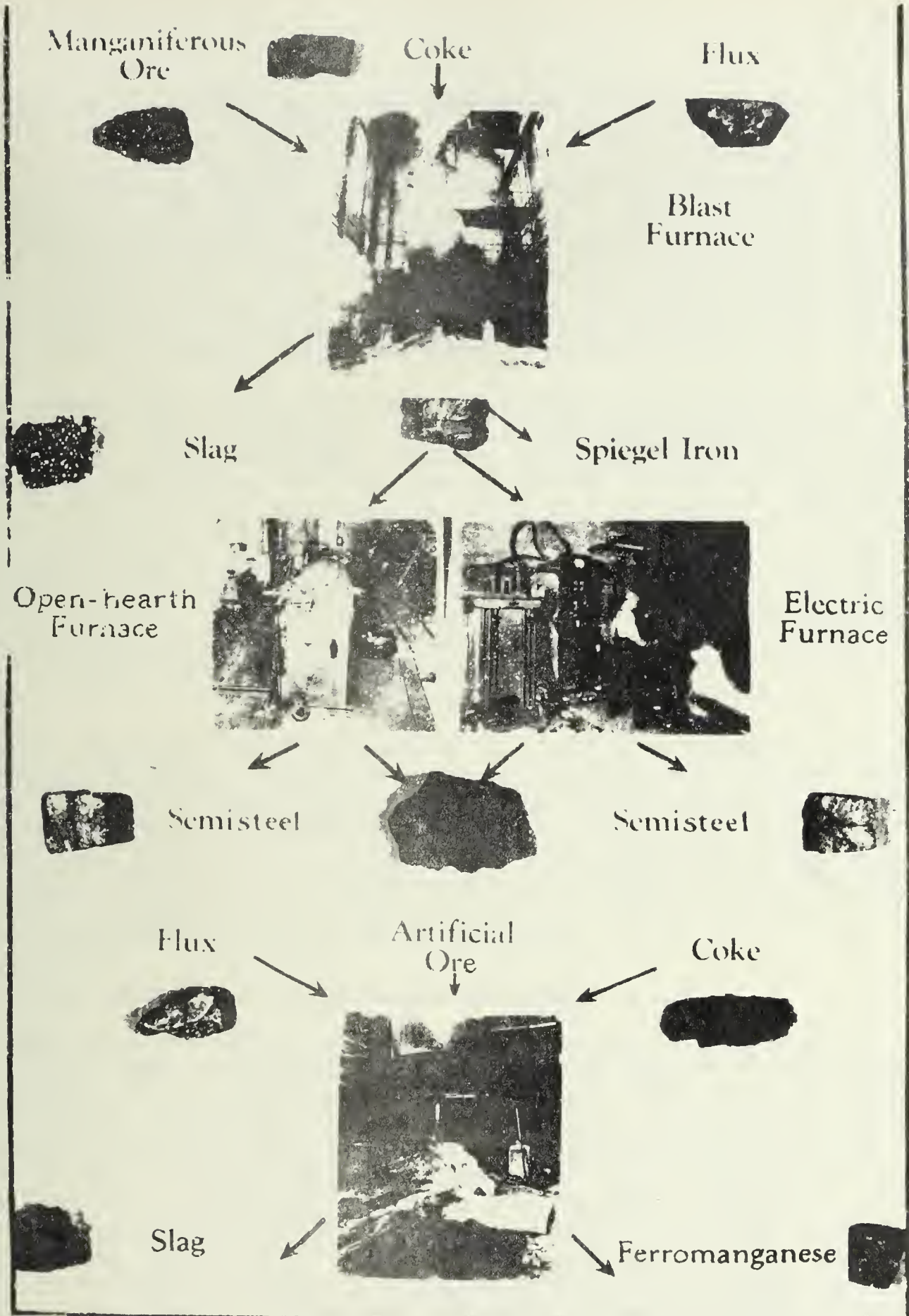


Figure 28.- Furnaces, raw materials, and products used in producing ferromanganese from manganese iron ores.

General data from the operation of the furnace are given in table 76. The furnace, in general, operated uniformly, and the stock settled regularly with no slip or top explosions which may be aggravated by the generation of large amounts of hydrogen in the top of the furnace. Percentage of hydrogen in the top gas averaged 2.23, which was not unusually high. There should be no difficulty in smelting the brown ores in industrial furnaces with an 80 percent recovery of manganese and a fuel consumption of 2,500 to 3,000 pounds of coke, depending upon the grade of coke, particular ore used, facilities for heating the blast, and practice in general. Careful grading of the ore to a uniform analysis would be essential to regular blast-furnace operation.

The manganiferous iron ores to which this process will apply contain about 50 percent iron plus manganese and 50 percent oxygen plus moisture and gangue. By eliminating the oxygen, moisture, and gangue in the blast furnace the manganese is increased from 8 percent in the ore to about 15 percent in the metal, and the iron is increased from about 40 to 80 percent. The percentage of phosphorus is also approximately doubled, increasing from 0.25 in the ore to about 0.5 in the spiegel. An alloy containing manganese, phosphorus, and iron in these proportions is not a commercial product, due chiefly to the high phosphorus. By separating the manganese from the phosphorus and the iron two usable products may be obtained; one, a high-manganese slag suitable for producing ferromanganese, and the other, a low-silicon and low-sulphur carbon-iron alloy that can be further refined into steel.

Newton (274) discussed the possibilities of treating manganiferous iron ores by duplex processes. Manganiferous iron ores high in phosphorus would first be converted to high-phosphorus spiegel in the blast furnace. It was proposed to blow this high-phosphorus spiegel in a basic Bessemer converter to convert the manganese into a slag which would be poured off before it was contaminated by phosphorus. Phosphorus was to be eliminated after the manganese in order to produce steel. This proposal was tested experimentally with the small side-blown converter shown in figure 29.

Results Obtained in Blowing High-Phosphorus Spiegel

Eleven blows were made on molten spiegel with equally unsatisfactory results in each blow. The blows were "sloppy", much slag being thrown from the converter. It was difficult to regulate the blast due to the large amount of slag produced. As shown in figure 30, the manganous oxide ranged from 60 to 65 percent during the first 12 minutes of the blows and then decreased to about 50 percent. Manganese was eliminated faster than phosphorus, but prohibitive amounts of the latter were thrown into the slag. The ferrous oxide in the slag ranged from 20 to 30 percent or was equal to one half to one third of the manganous oxide present. Manganese cannot be separated from phosphorus and iron to the degree required in producing artificial ferro-grade ore by differential oxidation in a Bessemer converter. A high-phosphorus alloy containing about 60 percent manganese could probably be made from basic converter slag resulting from bessemerizing high-phosphorus spiegel. The results in the small converter confirmed reported data (188) on two spiegel blows by Waterhouse in an acid-lined, bottom-blown, industrial converter. The slag from the large converter contained about three times as much manganese as iron. Since the objective was to obtain slag containing about ten times as much manganese as iron, no further converter tests were made.

TABLE 76. - General Data from Operation of Experimental Blast Furnace

	Period No.						
	1	2	3	4	5	6	7
1. Ore	6,547	6,015	5,460	5,831	5,704	5,628	6,472
2. Stone	2,078	1,744	1,456	1,079	1,407	1,398	1,312
3. Coke	3,805	3,707	4,368	3,594	2,961	3,106	3,162
4. Carbon	3,289	3,224	3,887	3,211	2,442	2,547	2,557
5. Metal per day.....	3.67	3.97	3.43	4.22	5.18	4.92	4.72
6. Slag	2,248	2,414	2,264	1,918	2,255	2,119	2,145
Ore analysis (nat.) .. percent							
7. Manganese	8.13	7.66	8.93	8.16	8.31	8.56	8.28
8. Iron	33.41	33.98	32.76	32.29	33.59	34.20	35.15
9. Silica	8.40	8.19	9.31	5.57	10.26	7.94	4.53
10. Alumina	6.45	6.54	6.01	5.98	5.23	6.14	5.25
11. Phosphorus	2.26	2.25	2.27	2.27	2.25	2.25	2.27
12. Moisture	15.23	16.21	16.84	21.36	16.50	16.85	18.28
13. Loss on ignition	9.65	9.00	10.17	10.45	9.94	10.08	11.38
Metal analysis							
14. Carbon	4.71	4.48	4.67	4.74	4.60	4.67	4.26
15. Manganese	11.97	11.96	15.33	13.82	11.44	13.05	10.96
16. Silicon	1.14	1.34	1.52	1.22	1.23	1.14	1.12
17. Phosphorus	0.59	0.59	0.61	0.57	0.51	0.47	0.53
18. Sulphur 1/							
Slag analysis							
19. Lime	40.24	33.81	38.34	33.88	37.75	39.64	36.97
20. Alumina	14.25	17.40	16.44	17.70	12.27	13.24	16.59
21. Silica	26.28	32.39	31.68	30.83	33.21	31.17	27.16
22. Manganese oxide	12.13	13.01	10.33	13.68	12.08	12.58	14.76
23. Ferrrous oxide CaO	1.78	1.38	1.79	1.09	1.21	1.88	3.33
24. Basicity (Al ₂ O ₃ + SiO ₂)	0.95	0.68	0.80	0.70	0.83	0.90	0.83
25. Mn charged	532	461	488	476	474	482	536
26. Mn to metal	50.5	58.0	70.4	65.2	54.1	60.7	45.7
27. Mn to slag	46.7	45.4	33.9	35.4	40.2	38.2	39.0
28. Mn unaccounted for 2/ ..	2.8	+ 3.4	+ 4.3	+ .6	5.7	1.1	15.3
Temperatures							
29. Blast	530	553	575	521	608	662	581
30. Top	825	704	906	695	446	579	535
31. Tuyère	1644	1676	1724	1693	1696	1716	1683
32. Slag	1335	1342	1476	1500
33. Metal	1370	1328	1342	1352	1313	1331	1406
34. Blast pressure	5.64	5.35	4.72	4.48	5.25	5.48	4.79
35. Blast pressure..lb. per sq. in.	2.77	2.63	2.32	2.20	2.46	2.69	2.35
36. Air per minute 3/	569	605	637	626	589	589	607

1/ Below 0.01 in all periods except No. 5, where three casts ran 0.02, rest below 0.01.

2/ Plus sign indicates an excess recovered over that charged.

3/ Calculated from carbon burned at tuyères and top-gas analysis.

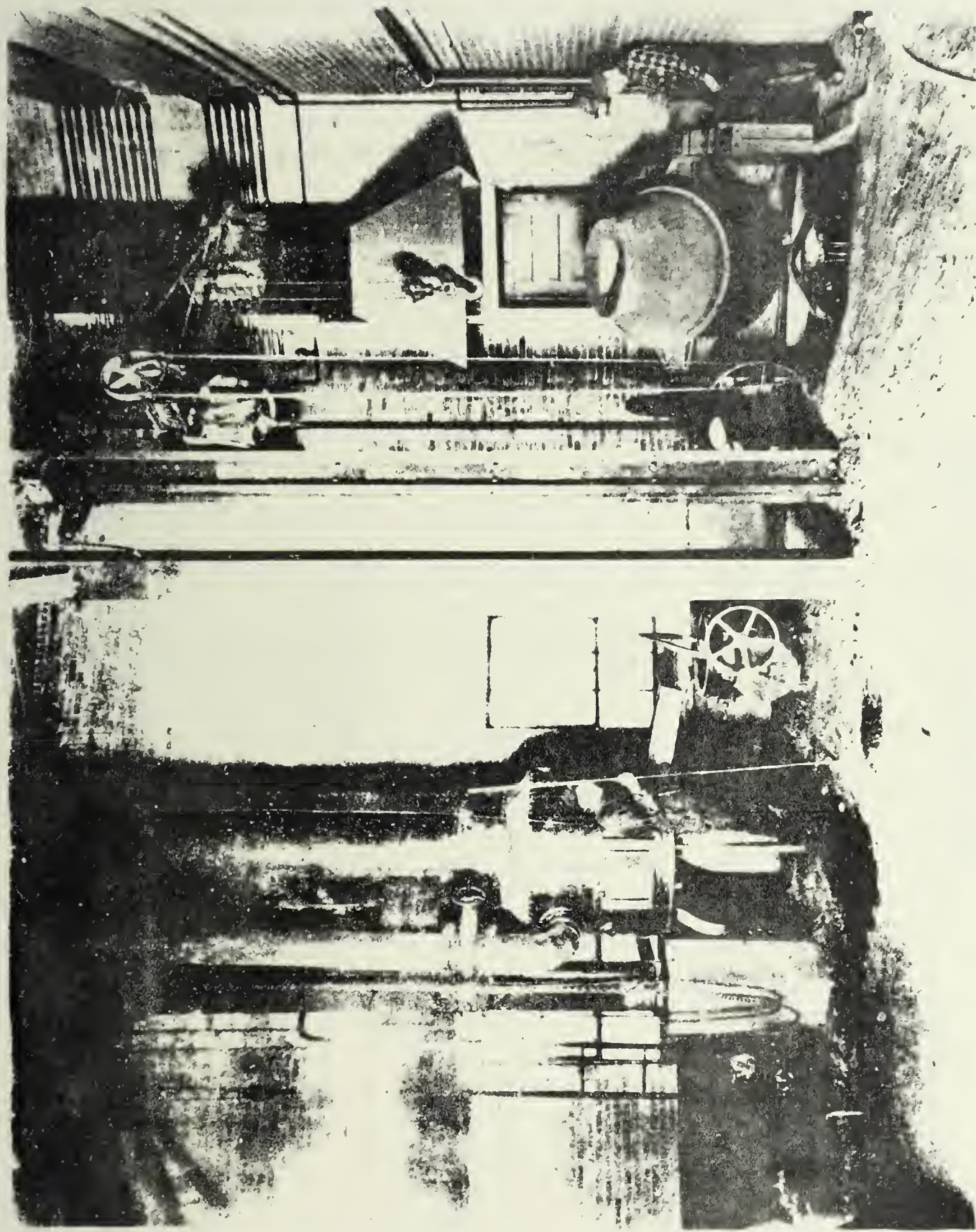


Figure 20 - Cuocla and experimental side-blow converter.



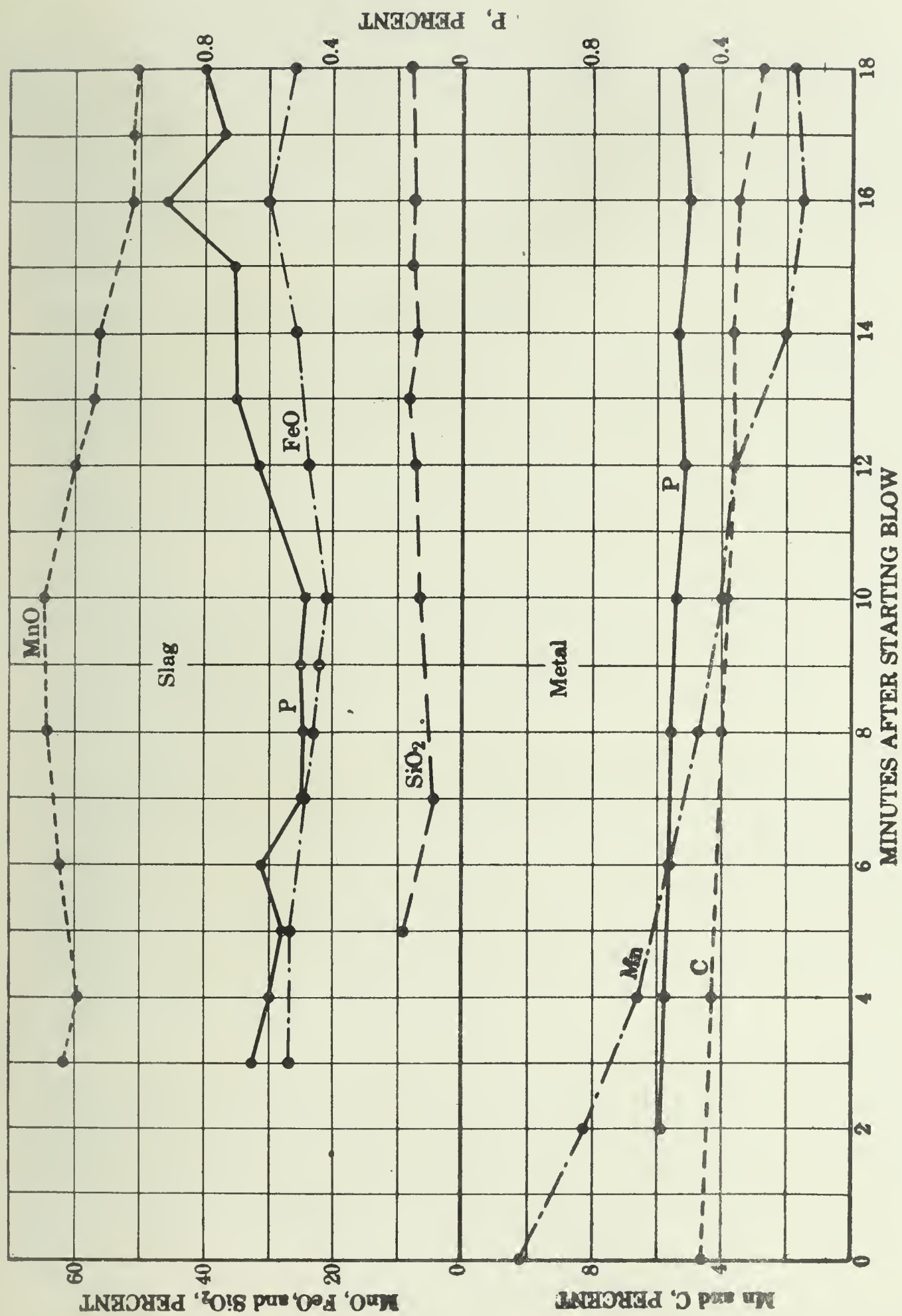


Figure 30.— Average change in slag and metal for 11 converter blows.

Tests in Acid-Lined Open-Hearth Furnace

Attempts were made to separate the manganese from the phosphorus and iron by oreing down charges of molten spiegel in a small acid-lined open-hearth furnace. The results were very unsatisfactory due to corrosion of the lining by the manganese and to prohibitive amounts of phosphorus, ferrous oxide, and silica in the slag. The acid slag contained as much as 1.5 percent phosphorus. Phosphorus is not thrown into the slag in acid open-hearth practice because such a small amount is present. Since the acid lining did not solve the phosphorus problem tests were made with a basic lining.

Tests in Basic-Lined Open-Hearth Furnace

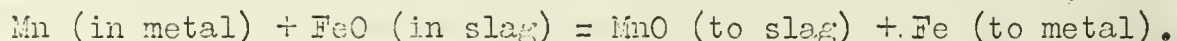
The difficulty of excessive corrosion of the furnace lining encountered in the acid open hearth was corrected by the use of a magnesite lining. This change in refractory lining did not solve the problem of producing slag containing about ten times as much manganese as iron and less than 0.2 percent phosphorus. If 4 or 5 percent manganese was left in the metal a proper ratio of manganous oxide to iron oxide in the slag could be obtained. Such amounts of manganese represented about one third of all of the manganese in the spiegel, so that such a procedure was not satisfactory.

The process was slow because with small ore additions a very thick slag was soon formed, and further action took place at an extremely slow rate. It became obvious that to eliminate the manganese rapidly and to get it down to 0.5 percent or under it was necessary to treble the amount of iron ore added. The fluid slag and high concentration of ferrous oxide forced the following reaction rapidly to the right, $\text{FeO} + \text{Mn} = \text{MnO} + \text{Fe}$.

The next problem was that of properly adjusting the ferrous oxide and phosphorus in the slag. Differences in the relative reducibility of ferrous oxide, phosphorus compounds, and manganous oxide suggested a means of adjusting the iron and phosphorus. Ferrous oxide reduces rapidly at 950° C. At somewhat higher temperatures (1,100° to 1,300°C.), phosphorus compounds are reduced, but it requires a temperature of about 1,400° C. and highly-reducing gases to deoxidize manganous oxide rapidly. Experiments showed that a 2- to 3-inch layer of small coke spread over the surface of the slag protected it from the oxidizing furnace atmosphere and provided condition under which the ferrous oxide and ferrous phosphate in the slag were reduced with a comparatively small reduction of manganous oxide.

Most of the experimental work on producing an artificial manganese ore from high-phosphorus spiegel was performed in the small tilting open-hearth furnace shown in figure 28. Tests with a small electric furnace showed that furnaces of this type have some distinct advantages over open-hearth furnaces for this particular operation (see p. 233). The procedure, which was essentially the same in both types of furnaces, can best be explained by following the progress of a typical open-hearth heat, shown in figure 31.

The furnace was first charged with 500 pounds of cold spiegel containing 15.75 percent manganese. If the process were conducted on an industrial scale the hot metal would be transferred directly from the blast furnace to an open-hearth or electric furnace. It was necessary to start with cold metal in most of this experimental work. Iron ore was added 2 hours after charging. As the manganese in the metal and the ferrous oxide in the slag decreased, the manganous oxide in the slag increased as a result of the following reaction,



Two and one-half hours after the ore was added or 4-1/2 hours after charging, the metal contained less than 0.5 percent manganese. The slag at that time contained 55 percent MnO, 32 percent FeO, and about 0.7 percent P. Coke was then added to adjust the phosphorus and the ferrous oxide in the slag. After holding the bath under a 2- to 5-inch layer of 1/4- to 1/2-inch coke for 2-1/2 hours the ferrous oxide was reduced to 6 percent. Meanwhile the manganous oxide had increased to 66 percent, which provided a 11:1 ratio of manganese to iron. The results of a great many heats show that the necessary adjustment of phosphorus in the slag occurs simultaneously with the reduction of ferrous oxide.

Detailed information on important features of experimental work in small open-hearth and electric furnaces has been published in several progress reports (189, 190) and a summary report (188). The important features of the work include the amount of ore added, composition of the slag, recovery of manganese, time required for operation, amount and composition of metal produced, and comparison between electric and open-hearth furnaces.

Composition of Slag

Results with small open-hearth furnaces indicate that two types of slag can be made. By using silica to maintain slag fluidity, slags can be produced containing 16 percent SiO₂, 65 to 70 percent MnO, 6 percent FeO, 0.16 percent P, 7 percent Al₂O₃, and a few percent MgO and CaO. The silica in the slag may be less than 10 percent, provided enough alumina is present so that the sum of the two constituents ranges from 19 to 23 percent. A typical slag in which part of the silica has been replaced with alumina would contain 65 to 70 percent MnO, 6 percent FeO, 0.16 percent P, 10 percent SiO₂, and 13 percent Al₂O₃. The amount of silica and alumina necessary to obtain a workable slag will depend upon the temperature and the advantage to be gained from shortening the time required for a heat.

Recovery of Manganese in Basic Open-Hearth Heats

The analysis of dust collected from metered portions of flue gas showed that about 1 percent of the manganese and 1.5 percent of the iron charged as spiegel were lost as dust. The ratio of manganese to iron in the dust samples was about the same as in the spiegel, indicating that the dust was produced by oxidation of the surface of metal globules thrown into the gas stream by the mechanical action of the bath. Samples of dust removed from blades of the exhauster and from the flues also contained manganese and iron in about the same ratio as the two occurred in the spiegel treated.

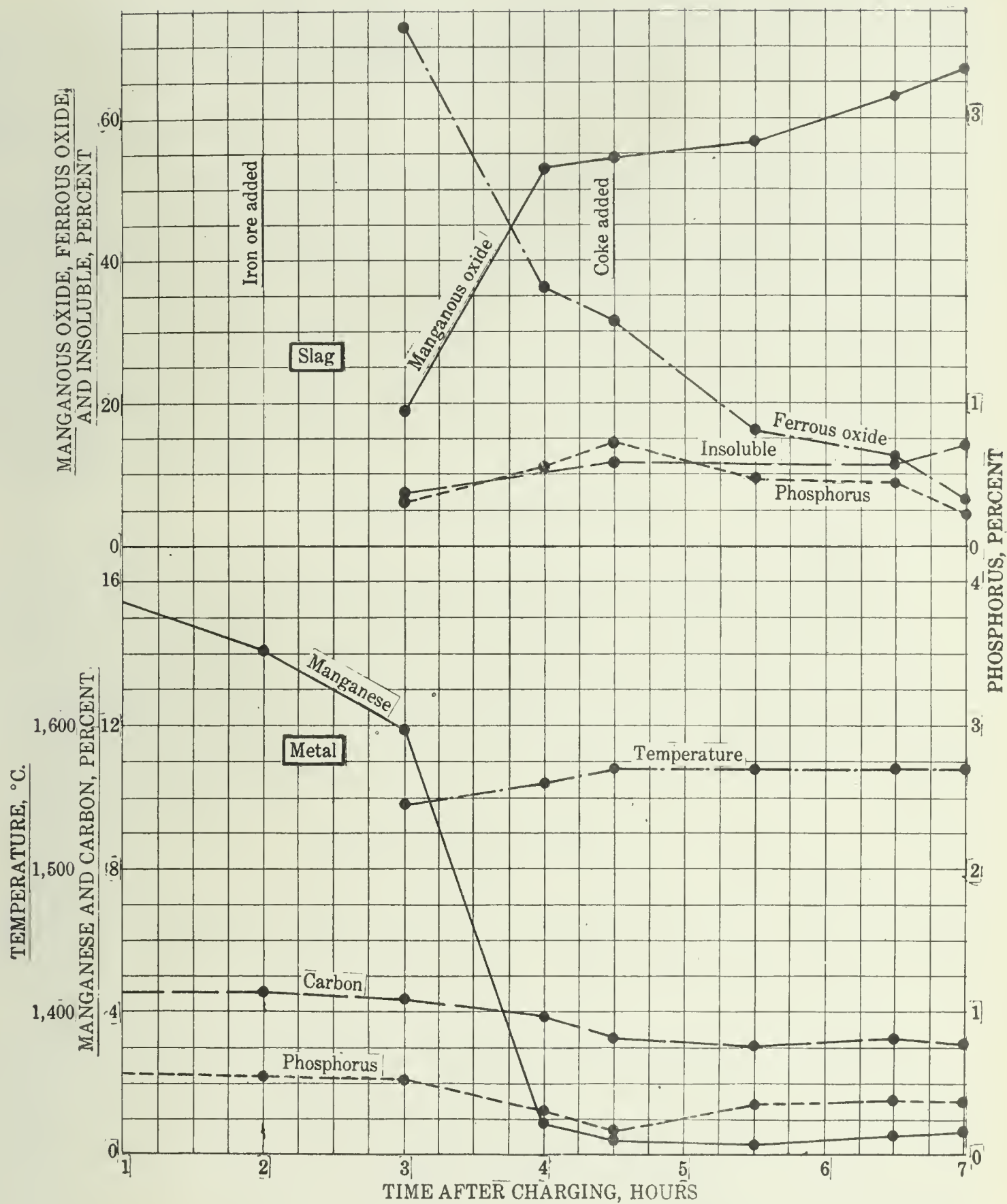


Figure 31.— Changes in composition of slag and metal in heat 437.

The data obtained from dust samples indicated that the loss of manganese by volatilization was small. This was confirmed by manganese balances on 200 open-hearth heats, which showed an average recovery of 35 percent of the manganese in the slag and 10.3 percent in the metal, leaving a loss of 4.7 percent.

Time Required for a Heat

The time required to recover the manganese in high-phosphorus spiegel in the form of a slag suitable for producing ferromanganese is governed largely by the length of time required to adjust the phosphorus and ferrous oxide in the slag. From 95 to 98 percent of the manganese was oxidized in an hour when the weight of iron in the ore addition was about 1.5 times the weight of manganese in the spiegel. The time required to adjust the phosphorus and the iron in the slag ranged from 1 to 3 hours. If the slag was fluid, the reduction of FeO and of $(\text{FeO})_3\text{P}_2\text{O}_5$ was rapid, provided the slag was completely covered with coke and was at a temperature of $1,575^\circ$ to $1,600^\circ \text{C}$. If the slag was viscous the adjustment of phosphorus and iron was slow and erratic. From 4 to 5 hours would probably be required for the complete operation in an industrial furnace.

COMPOSITION OF METAL PRODUCED

Approximately 1 ton of metal was produced for each ton of spiegel treated. The average analysis of metal produced in 16 typical heats was: Manganese, 0.87 percent; phosphorus, 0.43 percent; carbon, 5.4 percent; Sulphur, 0.002 percent; and silicon, 0.011 percent. In practice this metal would be refined into steel by the normal basic open-hearth process. The high-phosphorus content would be a detriment; but the extremely low sulphur content would be an advantage. Only through the value of this metal can the operation of recovering manganese from high-phosphorus spiegel be economical. The high-manganese slag should be regarded as a byproduct incidental to the production of steel from manganiferous iron ores.

Electric-Furnace Tests

High-phosphorus spiegel was treated in two stages, in the electric furnace and in the open-hearth furnace. The first stage was conducted under oxidizing conditions and the second under reducing conditions. Final adjustment of the slag was made in the last stage under reducing conditions.

The electric furnace has a decided advantage over the open-hearth furnace due to the more strongly reducing atmosphere that can be readily established. As a result of this advantage it required about one half as long to make heats in small arc furnaces as it did in the small open-hearth furnaces. In general, electric furnaces were more flexible than open-hearth furnaces. Being able to skim the slag manually from an industrial, circular electric furnace would be an advantage because slag lower in silica and alumina could be produced. Due to the shorter time required for the operation in the electric furnace it appears that the cost of treatment would be comparable to the cost of treatment in open-hearth furnaces.

Refractory Lining

A magnesite lining was satisfactory in both the open-hearth and electric furnaces. The bottom and sides tended to build up rather than to cut out.

Mineral Composition of Slags

The composition of the final slags in these tests is in that portion of the ternary system MnO-FeO-SiO_2 , proposed by Herty, Conley, and Royer (158), in which MnO is the major slag constituent. In the field adjacent to this knebelite (orthosilicate of Mn and Fe) is the major slag constituent. As a result of a rather preliminary petrographic study, McCaughey stated:

These slags are composed essentially of two minerals, manganosite and knebelite. In spite of the fact that the manganosite exceeds the silicate mineral in large amounts, the latter is the continuous phase and has crystallized out in fairly large crystals enclosing the manganosite.

Silicates of various metal oxides are reduced with more difficulty than oxides. It is important to note therefore that these slags are composed primarily of MnO and not of the silicate of manganese.

Production of Ferromanganese from
Artificial Manganese Ore

In producing slag containing 65 to 70 percent MnO and about 6 percent FeO from high-phosphorus spiegel, it is necessary to add a thinning agent or flux to obtain a degree of slag fluidity conducive to practical reaction rates. Herty, Conley, and Royer (158) found from tests on a crucible scale that alumina has a thinning effect upon high-manganese slag. They suggested its use in the production of high-manganese slag. Tests in a small open hearth and a small electric furnace showed that alumina could be used as a thinning agent. As stated on page 232, two types of slag can be made; one containing 16 percent SiO_2 and 4 to 6 percent Al_2O_3 , and another containing 8 to 10 percent SiO_2 and 10 to 12 percent Al_2O_3 . Although the high-silica slags appeared to be somewhat more fluid, satisfactory results were also obtained when using alumina as a thinning agent. Before drawing any conclusions, however, about the use of alumina as a substitute for part of the silica, it was necessary to investigate the smelting characteristics of the two types of slag.

Blast-furnace slag produced in smelting manganese ores normally contains about twice as much silica as alumina. If high-manganese slag, fluxed with alumina, were used as a manganese ore, the blast-furnace slag would contain about equal amounts of silica and alumina. Such a proportion of silica and alumina is rarely found in blast-furnace slag.

In producing pig iron the operator adjusts the slag composition to meet sulphur and silica specifications in the metal. In producing ferromanganese operators strive to adjust the slag so as to promote the complete reduction of manganous oxide. The slag in the crucible of a ferromanganese furnace

contains a series of compounds composed of CaO , MgO , SiO_2 , and MnO . In what manner or to what extent the proportion of silica and alumina changes the ease and completeness with which manganous oxide can be reduced from slag under blast-furnace conditions is not known. Herty, Conley, and Royer found from reduction tests in graphite crucibles at $1,550^\circ \text{C}$. that high-alumina slags reduced as readily as the high-silica slag.

For lack of a better working rule operators base the requirements for dolomite and calcite upon the amount of silica in the charge. Enough fluxing stone is added so that the sum of the lime and magnesia will be 1.5 to 1.9 times the total silica in the charge, including the coke ash. Bases are added to aid in the reduction of manganous oxide which is in solution in the slag and to limit the reduction of silica. Alumina is not reduced under blast-furnace conditions; therefore, the only question is whether basic oxides facilitate reduction of manganous oxide. If alumina in the charge can be ignored in calculating the amount of bases required to produce a slag from which manganous oxide can be reduced, it should be less objectionable than silica because of lower slag volumes. The effect of alumina upon pig-iron practice has been discussed, but no information appears to have been published on the effect of alumina on ferromanganese practice.

Tests in graphite crucibles by Herty, Conley, and Royer indicated that manganous oxide can be reduced satisfactorily from slag containing 40 percent CaO , 30 percent SiO_2 , and 30 percent Al_2O_3 on a three-component basis. Such slag offers encouraging possibilities from the standpoint of slag volume because of the unusually low basicity.

In view of the amount of work which had been done on the first two steps of the method proposed for producing ferromanganese from manganiferous ores, it was decided that the smelting characteristics of the two types of slag should be investigated with the experimental blast furnace. Such a test seemed desirable as it would complete on a small-furnace scale the last of a series of three operations by which it was hoped an alloy containing 80 percent manganese could be produced from ore containing 8 percent manganese.

EXPERIMENTAL BLAST FURNACE AS A TOOL FOR INVESTIGATING SMELTING PROBLEMS

The daily capacity of the experimental blast furnace is about one half of 1 percent of a modern industrial furnace. The question arises as to what types of problems can be investigated with such a furnace and what significance can be attached to the results obtained. The brief resume of former experimental furnace tests will illustrate its field of usefulness.

Segregating ore into sizes and charging the sizes into an experimental furnace in sequence improved the operation very markedly and suggested that within reasonable limits uniformity of size in a given charge has more effect upon the rate and completeness of reduction than does the absolute size of the average particle.

Prompted by the general suggestion (325) that uniform particle size in each layer should be beneficial, one plant has been separating its ore into

three sizes and charging each size separately for several years. This practice, by which production was increased 110 tons per day on foundry iron and 82 tons on basic iron with a corresponding coke savings of 438 and 437 pounds per ton, has been described by Kinney (205). Large variations in gas composition at various positions across several planes of the experimental blast furnace (327), forecast similar nonuniformity in full-size furnaces charged with bell and hopper, similar to the equipment used in the experimental furnace. Varying degrees of nonuniformity in temperature, in gas composition, and in gas velocity were found by Kinney (206, 207) in several industrial furnaces. Slags containing as high as 53 percent Al_2O_3 and as low as 6 percent SiO_2 have been melted and tapped without difficulty from the experimental blast furnace. The ease with which these slags were produced in the experimental furnace (185, 189, 191) left little doubt as to the possibility of producing in the blast furnace a slag similar in composition to the so-called high-alumina or quick-setting cements.

The experimental blast furnace is particularly useful in studying the smelting characteristics of new raw materials and in investigating ranges of slag composition outside of those of normal practice. The production of ferromanganese from two types of high-manganese slag presented a problem of this type.

The thermal requirements of the experimental furnace are vastly different from those of a large furnace due to large heat losses and low heat reserve of the little furnace below the tuyères. Experience on the small furnace shows that it requires 75 to 100 percent more fuel to do the same smelting work in the 1/2 percent unit as in the full-scale furnaces.

Raw Materials

Coke

Size of particle exerts a predominating influence upon the space required completely to convert atmospheric oxygen into carbon monoxide during the process of combustion. Normally, the oxygen and carbon dioxide are converted into carbon monoxide in localized zones extending 40 to 50 inches beyond the nose of the tuyères. If the coke particles range in size from 3/4 to 1-1/2 inches, the oxygen and carbon dioxide disappear in 18 to 22 inches. Byproduct coke, known on the market as "nut coke", was selected for the experimental blast-furnace test to minimize the size of the combustion zones and thus establish as strongly reducing conditions as possible in the hearth of the furnace. Typical percentage analyses of the coke follow:

Fixed carbon	Ash	Volatile matter	Sulphur	Ash			
				SiO_2	Al_2O_3	Fe_2O_3	CaO
90.67	8.38	0.95	0.53	--	--	--	--
92.30	7.37	.33	.55	--	--	--	--
91.93	7.59	.48	.42	35.21	27.2	28.22	6.12
91.37	7.81	.62	.57	35.72	29.6	28.33	3.12
91.00	8.00	.90	.50	37.10	26.0	26.00	6.20

FLUXES

High-grade calcite and dolomite were used as flux. In place of the 4- to 6-inch pieces used in industrial furnaces particles ranging from 1/2 to 2-1/2 inches were used. The distribution of various sizes is given in table 77. About 90 percent of the pieces ranged in size from 0.5 to 2.5 inches. Car samples showed the following percentage composition:

	CaO	MgO	SiO ₂	Al ₂ O ₃	Fe ₂ O ₃
Limestone	53.7	1.3	1.00	0.5	1.1
Dolomite	30.4	21.6	.80	.4	1.2

TABLE 77. - Screen Analyses of Raw Materials, Percent

Size	Coke	Calcite	Dolomite
-2.5 + 1.05 inches	76.5	52.1	46.7
-1.05 + .742 inch	21.0	28.6	32.7
- .742 + .525 inch	1.2	9.9	11.7
- .525 inch	1.2	9.4	8.9

	Artificial ore no. 11/	Artificial ore no. 22/	Manganese ore no. 1	Manganese ore no. 2
-2.5 + 1.05 inches	31.2	12.35	-	26.42
-1.05 + .742 inch	27.6	20.60	4.67	26.43
- .742 + .525 inch	18.2	14.45	6.54	14.95
- .525 + .371 inch	6.2	7.22	9.35	6.90
- .371 +3 mesh	4.9	9.28	11.20	4.60
-3 mesh +6 mesh	11.0	13.40	23.40	6.90
-6 mesh	-	22.70	44.84	13.80

1/ Silica higher than alumina. 2/ Alumina higher than silica.

High-Manganese Slags or Artificial Ores

Two types of high-manganese slag were used as a source of manganese. The greater part of the slag, which had been fluxed with silica, was produced by treating high-phosphorus spiegel in the open-hearth furnace. It was necessary, however, to make up virtually all of the high-alumina slag by remelting manganese ore in the open-hearth furnace and by adding bauxite as a source of alumina. The slag was tapped on top of a small amount of raw ore which aided in producing a more porous product.

Screen analyses of the two types of slag are given in table 77. The high-silica slag contained virtually no material smaller than 6-mesh. About 85 percent of it ranged in size from 0.4- to 1.5-inch pieces. The slag which had been fluxed with alumina was smaller in size (minus 14 mesh) but contained virtually no potential flux dust. From the standpoint of physical structure, material of this type is very desirable. Percentage analyses of the two types of slag follow:

	Mn	Fe	SiO ₂	Al ₂ O ₃	P
Slag fluxed with alumina	32.1	4.47	9.4	12.7	0.19
Slag fluxed with silica	49.8	4.50	16.0	5.0	.20

The slags are similar chemically except that in the first case about 7 percent silica has been replaced with an equal amount of alumina.

Similarity Between High-Manganese Slags and Sinter

The idea of using a fused product in place of an ore is neither new nor novel. It was formerly believed that a comparatively small amount of fusion occurred in sintering and that the formation of iron silicate was regarded as being highly undesirable. Klugh (308) states that the formation of iron silicates is due to high temperatures and can be avoided by proper control of time and temperature. More recently Schwartz (345) has shown by a microscopic study of polished surfaces that sinters are predominantly and in most cases almost entirely composed of a new generation of crystals or grains and that bonding is due to the interlocking of magnetic grains and the binding together of such grains by iron silicate. Sinter has more desirable physical structure than most iron ores. Its reducibility compared with iron ores has been the subject of considerable discussion and speculation and but little experimental inquiry.

High-manganese slag consists of manganous oxide bonded with iron and manganese silicates. It contains a large number of openings or pores formed by the evolution of gas. As in the case of iron-ore sinter, its resistance to size degradation aids in producing a more open blast-furnace charge comparatively free from dust. Complete and final reduction of manganous oxide is not accomplished before fusion occurs in any case. The fact, therefore, that a manganese-bearing product has been fused would not seriously affect its reducibility.

Commercial Ores

Two commercial ores were used in making up the synthetic slags. These were also used in the experimental blast furnace for a short period. As shown in table 77, one of the ores was rather fine, while the other had been crushed to a size comparable to that of the high-manganese slags.

Analyses of two ores, percent

	Mn	Fe	SiO ₂	Al ₂ O ₃	P
Manganese ore no. 1 ..	57.2	1.6	7.0	1.56	0.20
Manganese ore no. 2 ..	43.2	5.0	6.4	8.20	.19

These ores were not selected nor is their analyses reported with any implication that they are representative, commercial manganese ores. They were chosen primarily for convenience in making the synthetic slag.

DESCRIPTION OF EXPERIMENTAL BLAST FURNACE

A detailed description of the experimental blast furnace and of the plant layout has been published in a previous report (187). The dimensions and lines of the furnace are shown in figure 32. Normally, the weight of the slag is calculated. It was desirable to have more accurate information; therefore, the slag was caught in heavy cast-iron boxes. The ferromanganese and a portion of the slag were collected in one box and the overflow of slag in a second. This procedure avoided contamination of the slag by the ladle lining and permitted accurate weighing of the slag and the metal after removal from the boxes. In order to allow a longer time for reduction of manganese oxide in the crucible, the cinder notch shown on the left in figure 37 was not used, that is, the slag was not flushed from the furnace between casts. All of the slag produced was tapped with the metal at 3-hour intervals.

Discussion of Operating Results

Rate of blowing, slag viscosity, and burden or weight of raw material per unit of coke were factors for which there was no precedent or guide when the experimental blast furnace was blown in. It was necessary, therefore, to make a number of changes during the early part of the test. An air blast of 300 cubic feet per minute gave the best results. Faster rates of blowing led to increased slag losses, and at slower rates insufficient metal and slag were brought into the crucible to prevent the bottom of the furnace from building up. The slag and metal entering the crucible are the chief source of heat below the tuyère level. Experience with a number of small furnaces has shown that, unless the slag and metal are brought into the crucible at a sufficient rate to compensate for high heat losses and to provide a heat reserve below the tuyères, the crucible skull around the sides and the bottom of the furnace builds up.

Satisfactory operation of any furnace producing ferromanganese requires proper adjustment of slag basicity and of the amount of ore burden, which in this instance was high-manganese slag. The high-manganese slag, which had been fluxed with silica, was charged in the first half of the 30-day test (see p. 238 for analysis). By a series of changes in the burden and in slag basicity the manganese in the slag was reduced from 30 to 11.5 percent. Table 78 contains analyses of metal produced during the first part of the test.

The percentage of dolomite and calcite was varied, but time did not permit any systematic study of the effect of magnesia when operating on a charge containing normal proportions of alumina and silica. Higher percentages of magnesia gave better results on the high-alumina charges.

The results of the 30-day test have been summarized in table 79. Periods 1, 2, and 3 give the results obtained on the high-manganese slag or artificial ore which had been fluxed with silica. The most satisfactory results were obtained during period 1. The manganese in the slag averaged 11.5 percent and the basicity 1.59. Due to the large amount of iron introduced in the coke, the metal averaged 76.5 percent manganese, 1.5 percent less than the lower limit of 78 percent set for ferromanganese. The fuel consumption means very little unless it is considered in connection with past experience with the small furnace. Obviously, a furnace with a capacity of one half of 1 percent is much less efficient thermally than a full-scale furnace. From the test in which 12 to 15 percent spiegel was produced in the experimental furnace and from other tests with the same and smaller furnaces, it appears that 1 ton of ferromanganese could be produced in a large furnace from the same slag, with a coke consumption of 5,500 to 6,500 pounds.

TABLE 78. - Analyses of First 20 Casts of Ferromanganese,
Percent

Cast no.	Manganese	Silicon	Phosphorus	Carbon
1	77.0	0.05	0.41	6.5
2	79.5
3	82.8	.10	.43	6.4
4	82.5
5	81.6	.04	6.6
6	80.5
7	84.6	.11	6.2
8	82.3
9	83.4	.05	.41	6.5
10	82.0
11	80.3	.12	.42	6.6
12	84.9
13	87.9	.14	.43	...
14	83.1
15	77.71	.12	.43	...
16	78.4
17	80.7	.11	.28	...
18	78.4	.06
19	79.5
20	78.5	.10



TABLE 79. - Operating Data of Experimental Blast Furnace
During Ferromanganese Test

Item	Period			
	1	2	3	4
1. High-manganese slag lb. per ton	4,540	5,250	5,070	6,800
2. Calcite do	1,635	1,400	1,685	1,265
3. Dolomite do	1,635	1,400	1,685	1,265
4. Coke do	12,000	11,500	12,200	16,600
5. Carbon do	9,800	9,400	10,000	13,500
6. Slag do	4,350	4,558	5,258	6,400
<u>Metal analyses, percent</u>				
7. Manganese	76.5	74.0	74.0	73.0
8. Silicon1	.21	.12	.25
9. Carbon	6.5	6.5	6.5	6.3
<u>Slag analyses, percent</u>				
10. Silica	27.00	24.3	25.0	20.7
11. Alumina	11.65	10.7	11.0	17.7
12. Lime	31.80	23.7	28.5	19.0
13. Magnesia	12.15	9.1	11.2	6.2
14. Manganese	11.50	21.0	17.3	24.2
15. Basicity $\text{CaO} + \text{MgO}$	1.59	1.35	1.56	1.22
<u>SiO_2</u>				
16. Manganese charged lb. per ton	2,260	2,615	2,530	3,370
17. Mn to metal percent .	74.0	63.0	64.0	46.2
18. Mn to slag do	22.0	36.0	36.0	44.0
19. Mn unaccounted for do	4.0	1.0	.0	9.8
<u>Temperatures</u>				
20. Blast $^{\circ}\text{F.}$	930	860	942	720
21. Top do	1,200	1,177	1,000	1,032
22. Tuyere $^{\circ}\text{C.}$	1,740	1,700	1,735	1,710
23. Slag do	1,490	1,465	1,515	1,460
<u>Pressure</u>				
24. Blast in. Hg.	1.3	2.1	1.3	1.1
25. Blast lb. per sq. in.	.64	1.0	.64	.5
26. Air per min cu. ft.	300	300	300	300

There would be no difficulty in producing the standard 80-percent alloy because of the smaller amount of iron introduced with the coke ash. When a slag of lower iron content was used, it was possible to produce in the small furnace metal containing as high as 87.9 percent manganese (see table 78). The iron in the slag can be adjusted by varying the second step of the process by which the high-manganese slag is produced from high-phosphorus spiegel. With the fuel consumption that may be expected on full-scale furnaces, a ratio of manganese to iron of 10:1, that is, 50 percent manganese and 5 percent iron, would allow the production of the standard 78- to 82-percent ferromanganese.

Silicon and Phosphorus

The silicon in the metal produced in the experimental blast furnace was uniformly low and well under the 1 percent limit sometimes set for ferromanganese. It has been suggested that it would be difficult to control the silicon content of ferromanganese produced from high-manganese slag containing 15 to 16 percent silica. It is conceivable with a furnace operated primarily to produce tonnage and without due regard for recovery that control of silicon would present a problem when operating on high-manganese slag. The previous history of and the chemical composition and physical character of the raw materials used as a source of manganese have little bearing upon the reduction of silica from the slag below the fusion zone. With normal adjustment of basic fluxes and with fuel necessary to obtain a satisfactory recovery of manganese, there should be no difficulty in controlling the silicon in the metal.

The upper limit set for phosphorus in the slag was 0.2 percent, which would correspond to about 0.4 percent in the ferromanganese. Slag of lower phosphorus content (0.15 percent or under) could be produced. An addition of 14 pounds of metallic manganese in the form of ferromanganese containing 0.4 percent of phosphorus, would increase the phosphorus in the steel by 0.003 percent, which is about 0.001 percent more than is introduced when using an average grade of ferromanganese to make steel of average manganese content. Samples taken after making manganese additions and recarburizing often show a considerable increase in phosphorus content compared to samples taken from the furnace just before tapping. This increase of manganese is only partly due to the phosphorus introduced into the ferromanganese. A large part of the increase results from a kickback of phosphorus from the slag to the metal during the course of recarburization which establishes reducing conditions. Low phosphorus content is, no doubt, an aid in marketing ferromanganese, but its importance as an ingredient of this alloy has been overemphasized.

Although there are apparently no reasons why high-manganese slag could not be used alone in producing standard 80 percent ferromanganese, mixing with natural ores might be advantageous. The general practice is to mix various ores in proportions which give the best balanced mixture, physically and chemically.

Basicity of Slag

The most satisfactory results were obtained on the experimental furnace when slag basicity, expressed as $\frac{\text{CaO} + \text{MgO}}{\text{SiO}_2}$, was about 1.6. A more basic slag was produced, but the increase in slag volume more than compensated for the decrease in percentage of manganese in the slag. The loss of manganese in the slag is determined by the weight of slag and the percentage of manganese it contains. Periods 2 and 3 (see table 79) show that a lower percentage of manganese in the slag does not necessarily mean a smaller loss of manganese in the slag. The manganese in the slag averaged 3.7 percent less in period 3 than in period 2, but the loss in the slag was the same for both periods. It was very evident during the course of the test that when the slag basicity exceeded 1.6 to 1.7 the slag reaching the tuyeres was extremely basic due to some 25 to 33 percent manganous oxide. At such times comparatively cold lumps of limestone and slag reached the tuyère unmelted. The distance between the tuyères and the zone of fusion is related to the free-running temperature of the slag. Slag with a low free-running temperature melts at a higher level in the furnace, which means that it will trickle down over a longer coke column before reaching the tuyères. Although the time is comparatively short the surface contact is large. The temperature-viscosity relations of the slag influence the degree of spreading or dispersion of the slag as it drops down through the coke column against the rising gas stream. Basic oxides tend, in general, to retard the fusion of systems containing large percentages of manganous oxide and comparatively small amounts of silica. They also increase the viscosity. Hence, an excess of basic oxides, desirable in the last stages of reduction of manganous oxide in the crucible of a blast furnace, may be detrimental to the reduction of manganous oxide in and above the tuyère level.

Recovery of Manganese

The best recovery of manganese was obtained during period 1 when 74 percent of the manganese charged was recovered in the metal and 22 percent in the slag, leaving 4 percent unaccounted for. The large loss occurred in the slag and was due primarily to the high slag volume, which amounted to 4,350 pounds per ton of ferromanganese. With a fuel consumption of 6,000 pounds of coke, which might reasonably be expected on a large furnace, the slag volume would be reduced to about 3,500 pounds. This is about 1,000 pounds more slag than is normally produced when using the usual grades of imported ores. The larger slag loss resulting from this additional slag volume would be compensated for by a smaller stack loss due to the elimination of dusting. Measurements of the dust in the exit gases indicated that less than 1 percent of the manganese charged was carried off in the top gas. Manganese balances on periods 1, 2, and 3 showed a stack loss of from 0 to 4 percent, which merely means that the loss was probably on the order of 2 percent. The large loss indicated in period 4 will be discussed later. A stack loss of 10 percent and a slag loss of the same amount is a common distribution of manganese in furnaces operating on natural ores. By eliminating practically all of the dust loss and restricting the stack loss to a few percent loss by volatilization, the total smelting loss would not exceed 20 percent even though the slag loss were greater than 15 percent. Restricting the slag loss to 15

percent when producing 5,500 pounds of slag per ton of metal would require holding the manganese in the slag slightly under 10 percent. Such a degree of elimination of manganese from the slag is readily attained in good practice.

Fuel Requirements on High-Manganese Slag and on Natural Ores

More coke would be required to produce 1 ton of ferromanganese from high-manganese slag than from natural ores of lower silica content. Tonnage for a given furnace would be smaller because of higher fuel requirements and also due to the necessity of allowing greater time to realize the same degree of reduction of manganous oxide from the larger volume of slag.

The relation between slag volume and fuel consumption, reported by Royster to be 0.88 pound of carbon for each additional pound of slag, was discussed on page 210. On this basis, 1,000 pounds of additional slag per ton would require 880 pounds of additional carbon. Since 1 ton of ferromanganese can be produced from natural ores on a fuel consumption of 4,500 to 5,500 pounds of coke with normal blast temperatures, it seems reasonable to assume that the coke requirements in smelting high-manganese slag would range from 5,500 to 6,500 pounds.

Operating Temperatures of Experimental Blast Furnace During Ferromanganese Test

The calorized pipe stoves used to preheat the air placed a limit upon the temperatures to which the blast could be heated without burning out the pipe elements. Blast temperatures were therefore several hundred degrees lower than those usually maintained on industrial furnaces. Low blast temperatures were, no doubt, partly responsible for the high fuel consumption of the small furnace. Tuyère temperatures on the experimental furnace averaged $1,721^{\circ}\text{C}$. compared to $1,550^{\circ}\text{C}$. for full-scale ferromanganese furnaces. The temperature of the slag produced in the experimental furnace averaged $1,480^{\circ}\text{C}$. compared to $1,426^{\circ}\text{C}$. for industrial furnaces (see tables 73 and 79). It was particularly difficult to get accurate temperature measurements of the metal tapped from the experimental furnace, due to small casts and the short interval between the time when the metal was running freely and the time when the slag would start flowing. Metal temperatures, measured with an optical pyrometer, are often too high due to a small slag film on the surface of the metal. In general, the metal from the experimental furnace was cold. The lowest temperature observed was $1,325^{\circ}\text{C}$. Rapid dissipation of heat from the crucible of the small furnace is responsible for the large difference between the temperature at the tuyères and the temperature of the metal. This difference, amounting to 350° to 400°C ., corresponds to a similar value of 175°C . for industrial furnaces.

The low metal temperature, shallow crucible, small heat reserve below the tuyères, and overlapping of combustion zones are the principal factors which make a small blast furnace inefficient in producing ferromanganese. When the furnace was producing about one half ton of ferromanganese per day the bottom of the furnace remained at about the same level, the furnace was

easier to tap, and recoveries were higher. Important amounts of manganous oxide are, no doubt, reduced by contact between slag and metal. Larger quantities of hotter metal therefore aided in the recovery of manganese.

There is a marked change in reducing conditions in the tuyère zone and at lower levels. A shallow crucible does not provide enough storage space in which slag can be held under strongly reducing conditions.

Operation on High-Manganese Slag Fluxed with Alumina

The operation of the experimental furnace on high-manganese slag containing 49.8 percent Mn, 16 percent SiO_2 , and 4.9 percent Al_2O_3 , was much more satisfactory than the results obtained on high-manganese slag containing 52.1 percent Mn, 9.4 percent SiO_2 , and 12.7 percent Al_2O_3 . These two types of slag were very similar except for the ratios of silica to alumina, 3.3 and 0.74, respectively. The criteria for comparing furnace practice when producing ferromanganese are recovery of manganese, fuel consumption, and tonnage. By gradually adjusting slag basicity and the burden it was possible to produce from a high-silica charge slag averaging 11.5 percent manganese. With the prevailing slag volume, 74 percent of the manganese was recovered in the metal (see table 79, period 1).

When operating on a high-alumina charge the furnace did not respond to changes in slag basicity and the amount of fuel used. An effort was made during the course of 10 days continuous operation to get the manganese in the slag down to a reasonable figure. The best results and the only data which warranted summarizing and presenting in detail have been given in table 79, period 4. The manganese in the slag averaged 24.2 percent compared to 11.5 percent on the high-silica charge. Due to this high percentage of manganese 44 percent of the manganese was lost in the slag compared to a loss of 22 percent when operating on a high-silica charge. Only 46.2 percent of the manganese charged was recovered in the metal. About 10 percent of the manganese was not accounted for in the slag and metal. There are no apparent reasons for any greater volatilization loss or dust loss during the operation on the high-alumina charge. The slag produced during period 4 contained almost as much silica as alumina, whereas that produced during period 1 contained about 2.5 times as much silica as alumina. This difference in slag composition apparently affected the temperature-viscosity relations of the slag. The high-alumina slag contained considerable metal shot, ranging from 1/32 to 1/8 inch in diameter. In most cases these shot were concentrated in the lower part of the cold slag, which was collected in the same mold as the metal. This indicated that they were in suspension in the slag during tapping and settled out before the slag solidified after tapping. The poor separation of slag and metal indicated a difference in the physical properties of the slag. An effort was made to reclaim the larger shot, but there was an unavoidable loss of metal in the slag. Samples taken for chemical analyses were given a preliminary crushing and screened to remove shot which could not readily be removed, as they were feebly magnetic. Had it been possible to get a clean separation of slag and metal the recovery of manganese in the metal would probably have been about 5 or 6 percent higher, and the manganese unaccounted for would have been reduced to 4 or 5 percent.

There was a decided tendency for the bottom of the furnace to build up when producing the high-alumina slag. It was often necessary to turn the tapping bar upward in order to reach the liquid metal. At times it was necessary to open a hole at a higher level. This operating difficulty, as well as the poor separation of slag and metal, might be less pronounced on a large furnace, but the same troubles would probably occur to a modified degree.

The slag was frothy, full of gas, and porous after cooling, when it was low in manganese. This was observed particularly during period 1 (table 79). The high-alumina slags were dense, indicating that reduction of manganous oxide and gas formation were less active.

The numerous variables that affect the operation of a blast furnace often make it difficult to draw conclusions from operating data. In this particular case operation on high-alumina charge was sufficiently different from the operation on the high-silica charge to leave little doubt as to a difference in smelting characteristics of the two materials. The rather sharp increase in the percentage of manganese in the slag when the high-alumina charge reached the crucible is shown in table 80, which contains analyses of the last slag produced during period 3 and the first flushes of period 4. There was very little change in tuyère temperature during the 48-hour period. In fact, the tuyère temperatures were uniformly high throughout the entire test. The amount of burden was held constant. Blast temperatures were somewhat lower during the operation on high-alumina charges, but in view of the comparative small difference in tuyère temperatures the difference in blast temperatures does not explain the large difference in the operation on the two types of material. The capacity of the crucible below the tuyères was materially decreased when operating on the high-alumina charge. This was due to the incrustation of slag on the bottom and sides of the crucible.

TABLE 80. - Slag Analyses, Showing Transition from Normal to High-Alumina Slag, Percent

Samples taken 24 hours before high-alumina charge reached crucible

Percent				
Mn	SiO ₂	Al ₂ O ₃	CaO	MgO
12.88	27.7	11.2	28.0	9.40
15.9	24.8	11.1	28.2	9.70
15.3	22.6	10.8	30.25	12.31
17.4	23.02	10.7	30.2	12.20

Samples taken 24 hours after high-alumina charge reached crucible

22.3	26.16	11.38	18.3	9.11
26.8	20.00	13.00	14.2	7.06
25.7	19.32	17.3	17.2	7.03
26.4	18.7	17.7	19.7	6.95

As a result, the slag appeared in front of the tuyères in a comparatively short time after cast. This decrease in the volume of slag which could be held in the crucible below the tuyère level, where a highly-reducing atmosphere prevailed, aggravated the difficulty of being able to reduce the manganous oxide from the high-alumina slags.

Effect of Slag Basicity and Magnesia on Recovery
of Manganese from High-Alumina Slags

The first step in the method proposed to produce ferromanganese from manganiferous iron ores consisted in smelting a charge of such ore in the experimental blast furnace. During this test, the alumina in the slag ranged from 9.4 to 21 percent. At that time an effort was made to determine the effect of alumina on the completeness with which manganous oxide was reduced from the slag. The conclusion reached was that as the alumina in the slag increased the use of additional bases had less effect in decreasing the manganese in the slag; that is, there was a greater decrease in the manganese in the slag for a given increase in basicity when the alumina averaged about 10 percent than when it varied up to 20 percent. This same tendency was more pronounced during the production of ferromanganese from high-alumina charges. The alumina in the slag made during the production of high-phosphorus spiegel was almost as high as it was in the slag made during the ferromanganese test. There was, however, a large difference in the relative percentage of silica and alumina in the slag made in the two operations. The spiegel-slag contained from 2 to 2.5 times as much silica as alumina, whereas the ferro-slag contained approximately equal amounts of these two oxides.

In attempting to reduce the manganous oxide in the slag during the ferro-manganese test, the basicity was varied from 1.22 to 1.65. As shown in table 81, the manganese in the slag was practically the same when the basicity was 1.22 and 1.65. During the 10 days operation on high-alumina slag there was rarely less than 20 percent of manganese in the slag. The burden throughout this period was the same as during period 1, when the manganese in the slag averaged 11.5 percent.

Increasing the amount of magnesia in the slag was more effective than increasing the basicity in aiding the reduction of manganous oxide from the slag. For example, in table 81 a definite relation exists between the magnesium oxide in the slag and the percentage of manganese.

TABLE 81. - Average Slag Analyses Showing Effect
of MgO on Percentage of Manganese

Group	Percent					Basicity
	Mn	MgO	CaO	SiO ₂	Al ₂ O ₃	CaO + MgO
						SiO ₂
1	19.2	11.0	21.3	20.2	17.35	1.59
2	24.4	7.9	20.6	18.2	17.20	1.65
3	24.2	6.2	19.0	20.7	17.70	1.22
4	26.5	5.9	20.5	18.3	17.20	1.45
5	28.4	2.7	21.4	15.7	16.20	1.53

The percentage of manganese in the slag, however, was about the same when the basicity was 1.65 in group 2 as it was when the basicity was 1.22 in group 3. McCaffery, Cesterle, and Fritsche (248) have studied the effect of magnesium oxide upon the viscosity of the quaternary system $\text{CaO} \cdot \text{SiO}_2 \cdot \text{Al}_2\text{O}_3 \cdot \text{MgO}$. Their results show that magnesia has a pronounced effect towards increased fluidity of silica-alumina-lime-magnesia slags. The slags produced in the experimental furnace on high-alumina charges contained 25 to 35 percent manganous oxide. Such large percentages of a fifth constituent leave some doubt about applying viscosity measurements made on a four-component system. It appears probable that the higher percentages of magnesia did aid reduction of manganous oxide by decreasing the viscosity of the slag. The amount of trapped shot in the slag as indicated by the total iron determinations was less with higher percentages of magnesia. This indicates magnesia had a thinning action on the slag. Magnesia, however, was more of a palliative than a cure for the troubles encountered with high-alumina slag.

Operation of Experimental Blast Furnace on Manganese Ores

The experimental furnace was operated for a short time on two manganese ores. Ore 2 was high in alumina and the other contained silica and alumina in normal proportions (see p. 238). The time available was too short to correct channeling of the gas, which was particularly bad on ore 1. The analysis of metal produced from manganese ore is given in table 82.

Commercial furnaces, which have a comparatively large amount of reserve heat in the hearth or crucible, are much less sensitive to changes and irregularities than the small furnace. Nonuniformity in size of raw materials and the inevitable nonuniformity in gas distribution, however, tend toward inefficient operation in large furnaces.

Temperature surveys were made across several planes of the experimental blast furnace to obtain a comparison in gas distribution when operating on the high-manganese slag and when operating on natural ores. Figure 33 shows the position of the test holes. The observations are reported in table 83 and in figure 34. All observations made across various planes when the furnace was operating on high-manganese slag showed uniform increases in temperature from the inwall toward the center of the furnace.

Temperature measurements made through hole 64 in plane 6 on June 4 were very similar to those made through the same hole on June 9. A striking change is shown between the gradual temperature changes on these days and the sharp fluctuation of temperatures with distance from the inwall when the furnace was filled with fine ore. The physical characteristics of the slag or artificial ore were important enough in the operation of the small furnace to offset the advantages of lower silica in the natural ores. This would probably not be true of a large furnace. The physical structure of the slag, which resembles sinter, would, however, offer an advantage in establishing uniform distribution of the gas, which is essential to uniform preparation of the material in the upper part of the furnace.

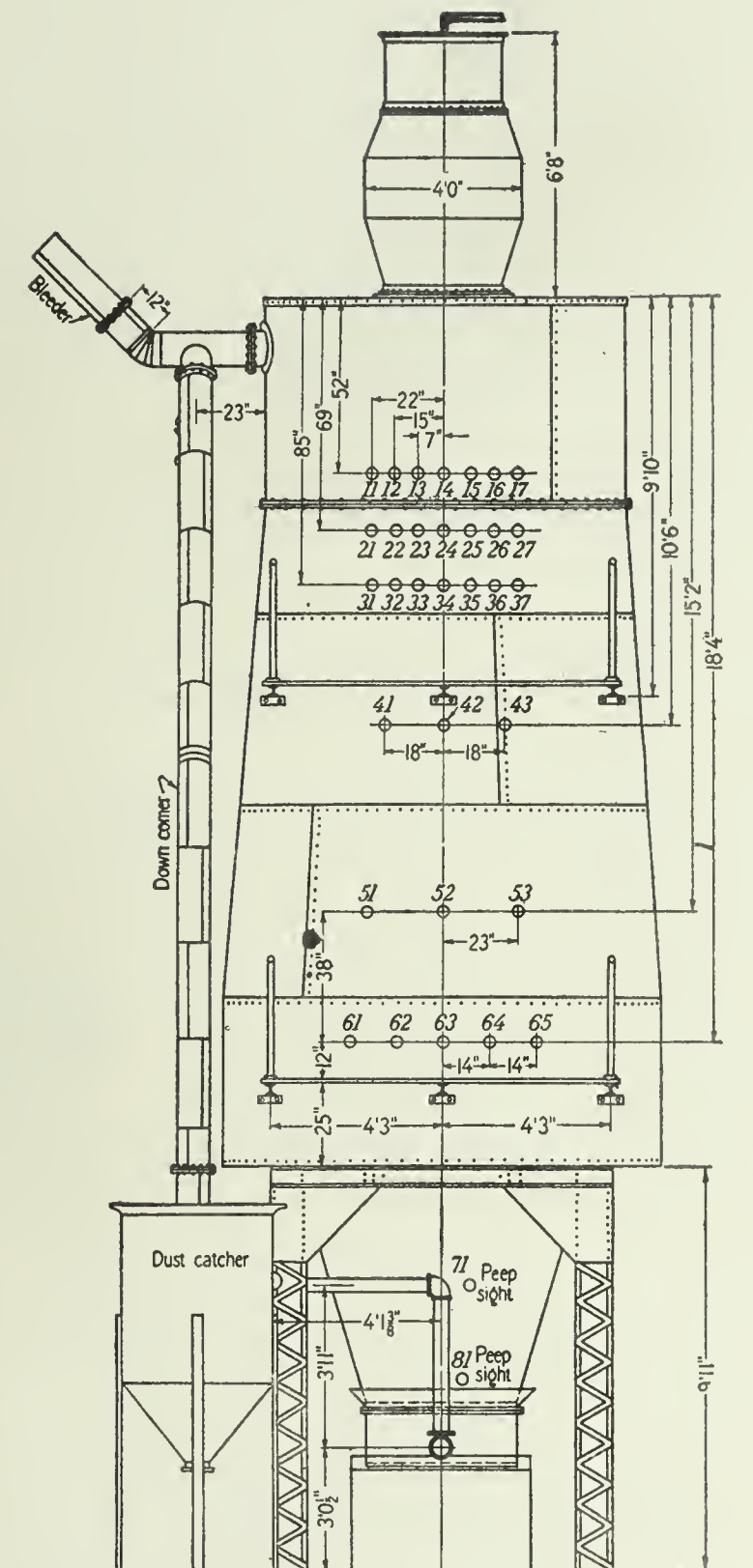


Figure 33.- Position of holes in lining of experimental furnace.

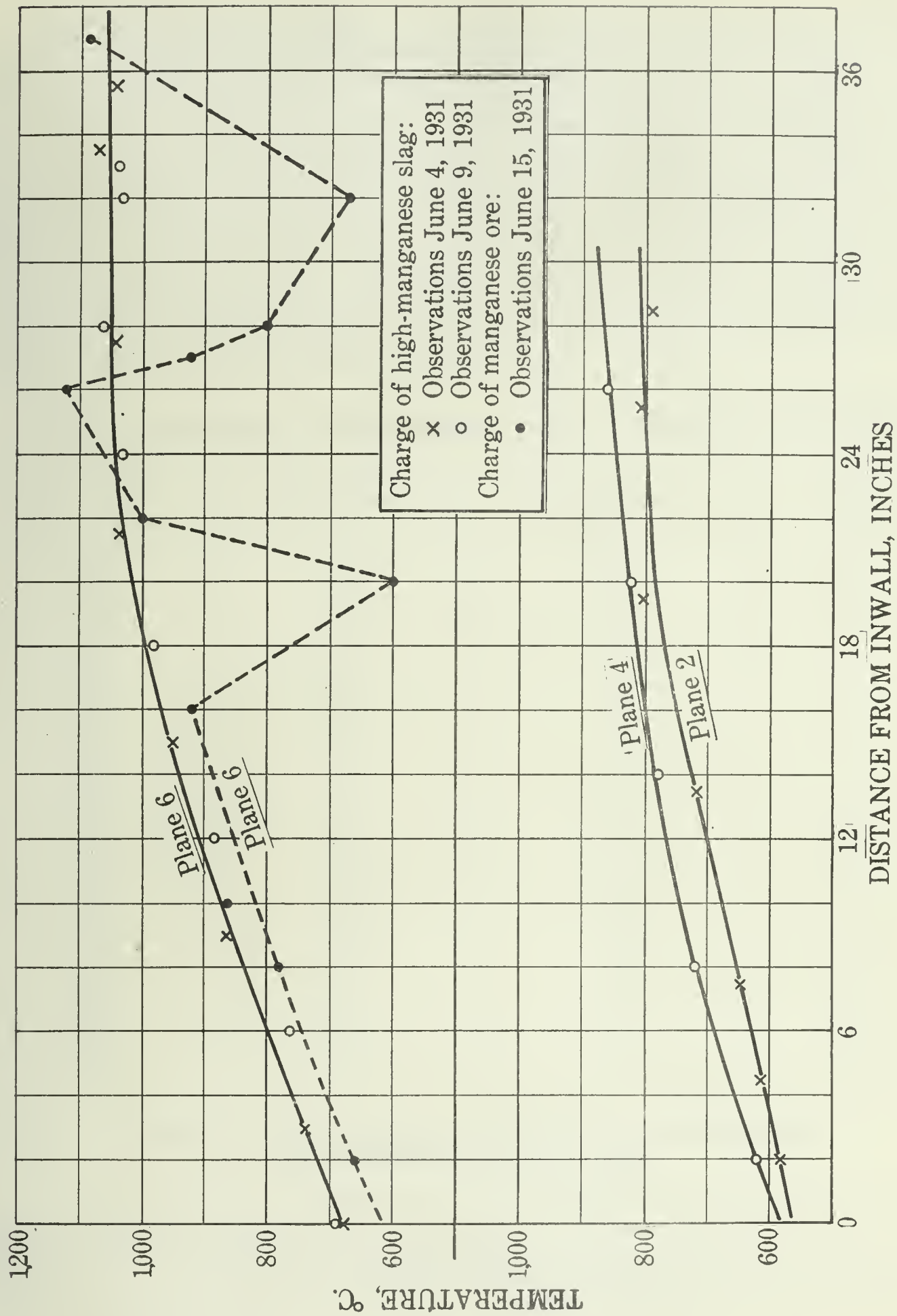


Figure 34.-- Temperatures along three planes of experimental blast furnaces.

The short period of operation on natural ores was significant in one other respect. Ore 2 was used first, being followed with a charge of ore 1, which was finer and apparently settled through the furnace more rapidly. About 4 percent more manganese was recovered in the slag and metal than was charged in ore 2, indicating that substantial amounts of ore 1 had worked down through the furnace at a rate more rapid than the normal rate of stock descent. A manganese balance for the period of operation on ore 1 showed a stack loss of 31.8 percent. Considering the entire period on natural ores, the stack loss averaged 27.8 percent compared to an average stack loss of about 4 percent for operation on high-manganese slag. Such a large difference in stack loss would not probably be found if the two types of material were smelted in large furnaces. The data, however, from the small furnace indicate that much of the so-called stack loss is a dust loss which may vary widely with the ore used. The physical structure of high-manganese slag would practically eliminate this dust loss.

TABLE 82. - Analyses of Ferromanganese Made from Manganese Ore, Percent

Cast no.	Manganese	Phosphorus	Silicon
203	79.7	0.43	0.05
204	76.9	----	----
205	80.1	.40	----
206	76.4	----	.26
207	78.7	.34	----
208	79.3	----	----
209	75.4	.40	.10
210	76.6	----	----
211	76.6	.34	----
212	78.0	----	.08
213	76.6	.42	----
214	75.8	----	.59
215	73.01	.43	----
216	79.46	----	.02
217	76.10	.47	----
218	80.10	----	.02
219	75.8	.39	----
220	77.7	----	.06
221	76.6	.41	----
222	73.8	----	.36
223	78.0	----	.39

Summary of Experiments on Production of Ferromanganese
from Manganiferous Iron Ores

The importance of manganese in the production of steel and the comparatively small domestic reserves of manganese ores have focused attention upon the problem of producing ferromanganese from our more abundant deposits of manganiferous iron ores. The experimental work was centered upon the brown manganiferous iron ores which occur in large well-developed deposits on the Cuyuna range in Minnesota.

TABLE 83. - Temperatures Across Three Planes of Experimental Blast Furnace During Ferromanganese Test

Hole 23, June 4, 1931

Distance from inwall, inches	Temperature, °C.
2.0	585
4.5	615
7.5	650
13.5	720
19.5	805
25.5	810
28.5	790

Hole 43, June 9, 1931

2.0	620
8.0	720
14.0	780
20.0	820
26.0	860

Hole 64, June 4, 1931

0.0	685
3.0	740
9.0	865
15.0	950
21.5	1,035
27.5	1,040
33.5	1,070
35.5	1,042

Hole 64, June 9, 1931

0.0	690
6.0	760
12.0	880
18.0	980
24.0	1,030
30.0	1,060
34.0	1,030
35.0	1,035

Hole 64, June 15, 1931

2.0	660
8.0	780
10.0	860
16.0	920
20.0	600
22.0	1,000
26.0	1,120
27.0	920
28.0	800
32.0	670
37.0	1,085

Tests with an experimental blast furnace, a small open-hearth furnace, and small electric furnaces of the arc type have demonstrated that ferromanganese can be produced from manganiferous iron ores by a three-step process, as follows:

1. The ores are smelted in a blast furnace according to the general practice of producing spiegeleisen. The metal produced does not, however, conform to standard grades of spiegel. It would contain from 12 to 15 percent manganese, 0.5 percent phosphorus, and about 80 percent iron. There is no market for metal containing manganese, iron, and phosphorus in these proportions.

2. High-phosphorus spiegel, made in step 1, is next treated in either a basic open-hearth furnace or a basic electric furnace. Iron ore is added to oxidize the manganese and to get it into the slag. To do this rapidly and with the desired degree of completeness an excess of iron ore is used. Phosphorus is oxidized simultaneously with the manganese and enters the slag. There is enough difference in the rate of reduction of ferrous phosphate, ferrous oxide, and manganous oxide to permit adjusting the iron and phosphorus with a comparatively small return of manganese to metal. The necessary reducing conditions can be established by covering the slag with a 2- or 3-inch layer of small coke. When the slag contains 25 to 35 percent ferrous oxide it is very fluid, but as the ferrous oxide content is lowered to 6 or 8 percent and the manganous oxide is increased to 60 to 70 percent the slag becomes very viscous unless silica or alumina is added as a thinning agent. It would be feasible to add predetermined amounts of these oxides at the beginning of the heat or to use iron ore containing amounts of silica and alumina needed to obtain a workable slag. Although alumina has a thinning effect upon the slags, it is neither as abundant nor as cheap as silica. If some substantial advantage could be gained in the next operation of converting the high-manganese slag or artificial manganese ore into ferromanganese, the use of alumina might be justified.

The electric furnace has distinct metallurgical advantages over the open-hearth furnace for separating the manganese from the iron and phosphorus in low-grade spiegel. The chief advantages are in flexibility and in a more strongly reducing atmosphere which permit greater speed in adjusting the iron and phosphorus to the limits usually set for manganese ore.

3. High-manganese slag containing 60 to 70 percent manganous oxide, 5 to 6 percent ferrous oxide, and about 20 percent silica plus alumina resembles sinter, another fused product. It is porous and contains manganosite as a principal slag constituent. There was no serious doubt about being able to produce ferromanganese from high-manganese slag, but there was evidence from crucible tests indicating that alumina has distinct advantages over silica as a thinning agent. It was desirable to check this evidence with an experimental furnace test. Information obtained upon this point and upon other phases of ferromanganese practice follows:

- (a) Operation of the experimental blast furnace on high-manganese slags fluxed entirely with silica was more satisfactory than on slags fluxed with alumina. In view of the greater cost of alumina as compared to silica and of the results obtained in smelting the two types of slag in the experimental furnace, alumina is not recommended as a thinning agent for high-manganese slag intended for use in the production of ferromanganese.

(b) High-manganese slag resembles sinter in physical texture. It is porous, contains little potential dust, and undergoes no size degradation in the furnace. Dust losses were almost completely eliminated in operating the experimental furnace on this material. The total loss of manganese due to dusting of the ore, volatilization of manganese, and oxidation of the manganese at the tuyeres followed by dusting ranges from 15 to 25 percent in practice. Some manganese ores result in high stack losses because they become very soft after heating and break down to dust under the abrasive and crushing action of the coke in the blast furnace. Size degradation in the furnace not only leads to dust losses but causes channelling of the gases. The gas distribution in the experimental furnace was far more regular when operating on the high-manganese slag than when operating on two natural manganese ores, one of which was particularly susceptible to dusting. Due to the small amount of dust produced when operating the experimental furnace on high-manganese slag, it was possible to isolate the loss of manganese by volatilization. This loss, not distinguishable from ordinary dust losses in commercial practice, averaged 4 percent of the manganese charged. The physical character of the slag, which was particularly beneficial in operating the small furnace, would also be desirable in the operation of industrial furnaces.

(c) High-manganese slags are less desirable chemically than the imported ores from which most of our ferromanganese is produced. Silica, essential in the second step in maintaining a fluid and workable slag in the presence of 60 to 70 percent manganous oxide, would increase the weight of slag per ton of ferromanganese about 1,000 pounds over the amount normally produced. This additional slag would, if it contained 10 percent manganese, represent a loss of about 5 percent of the manganese charged. Such a loss would probably be offset by eliminating dust losses. The 1,000 pounds of additional slag would require additional coke and would decrease furnace output.

(d) Results from the experimental furnace indicate that it is desirable to use part dolomite as flux. This is particularly true as the amount of basic fluxes is increased. Slag reaching the tuyeres and crucible contain manganous oxide which together with calcium oxide and magnesium oxide -- essential in the final stages of reduction -- tends to increase the viscosity of the slag. It appears that magnesium oxide promotes fluidity and is particularly valuable, while the slag is still high in manganous oxide. Equal amounts of dolomite and calcite gave the best results on the experimental blast furnace.

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MANGANESE; ITS OCCURRENCE, MILLING, AND METALLURGY

PART IV.

CHAPTER 9. FERROUS ALLOYS OF MANGANESE AND THEIR USE IN THE
STEEL INDUSTRY



BY

G. R. FITTERER AND M. B. ROYER

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UNITED STATES BUREAU OF MINES

MANGANESE; ITS OCCURRENCE, MILLING,
AND METALLURGY¹

PART IV

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1. The first part of the document discusses the importance of maintaining accurate records of all transactions and activities. It emphasizes the need for transparency and accountability in financial reporting.

2. The second part of the document outlines the various methods and techniques used to collect and analyze data. It includes a detailed description of the experimental procedures and the statistical analysis performed.

3. The third part of the document presents the results of the study. It includes a series of tables and graphs that illustrate the findings of the research. The data shows a clear trend of increasing activity over time.

4. The fourth part of the document discusses the implications of the findings. It suggests that the results have significant implications for the field of research and may lead to further developments in the future.

5. The fifth part of the document concludes the study. It summarizes the main findings and provides a final statement on the importance of the research.

CHAPTER 9. FERROUS ALLOYS OF MANGANESE AND THEIR USE
IN THE STEEL INDUSTRY

By G. R. Fitterer² and M. B. Royer³

HISTORICAL REVIEW

The application of manganese in iron and steel metallurgy is very interesting historically. In general, ancient irons and steels contained virtually no manganese, probably because the available iron ores contained only small amounts of the manganese oxides. Occasionally, samples of ancient irons were found to have appreciable amounts of manganese, but their presence is considered accidental and dependent upon the ore available to craftsmen of that time. Two specimens of iron (139)⁴ made in 500 A.D. contained negligible quantities of manganese. A sample of steel which constituted the base of the Khan Baba pillar in India was probably processed in 125 B. C. by the cementation of wrought iron. This material contained 0.02 percent manganese and 0.70 percent carbon (20). One Damascus-steel sword forged some time between 1160 and 1400 A. D. was found to contain 0.14 percent manganese and 0.61 percent carbon. The manganese content of this blade was much higher than that of any other material of this type which has been analyzed. In every other instance the manganese content was below 0.02 percent.

A cast steel manufactured by the East India Co. in India in the early nineteenth century, under the name of Wootz steel, contained rather appreciable amounts of manganese. This material was made by melting and refining fairly high-manganese pig iron prepared from high-manganese ore. Heath (106), who was working at the Co. plant between 1820 and 1840, was convinced that the manganese content of the steel was the determining factor in its high quality. He learned of certain difficulties English steel manufacturers were having in making even fair-grade steel, and analyses of the material indicated low manganese contents. This convinced him that if the use of manganese could be introduced in England its steel would be greatly benefited. In consequence, he moved to England and soon was producing certain grades of ferromanganese for use with the crucible process. He recommended addition of this metallic material to the ingot mold just before pouring. It was soon found that inferior brands of steel containing only traces of manganese could be greatly improved by such additions. All metallurgists of that time became convinced

² Former Associate metallurgist, U. S. Bureau of Mines.

³ Assistant metallurgist, U. S. Bureau of Mines.

⁴ Throughout parts I to IV, numbers in parentheses refer to items in this bibliography, and page numbers to those in the reference cited.

that manganese increased the weldability and malleability of iron.

Heath patented the use of ferromanganese in crucible-steel practice but later stated that the same effects could be produced by adding manganese ore and charcoal to the steel in the crucible. This procedure was found to be satisfactory and was used instead of his patented process.

Bessemer (106) then became interested, through his contacts with Heath, in the possibilities of increasing the quality of the product of his converter by means of manganese additions. Bessemer prepared ferromanganese by reducing iron and manganese oxides with charcoal in the crucible furnace. At the time of Bessemer's experiments Mushet (106) patented a triple alloy of manganese, iron, and carbon and recommended the use of this material as an addition agent in the bessemer process. This patent was ignored after several years because it was found that such material was already in use in special forms of high-manganese pig iron from Scotland and spiegeleisen (containing about 8 percent manganese) from Germany. The manganese content of these materials had not been previously considered in this connection. Bessemer continued his experiments on the preparation of ferromanganese and interested a chemist named Henderson (106) in the manufacture of a high-grade ferromanganese. Bessemer was particularly interested in obtaining an alloy containing about 75 percent manganese and 2 to 3 percent carbon, so that it could be added to the low-carbon steels made by the bessemer process without appreciably raising the carbon content. Henderson was able to make such material, and the use of the bessemer converter was greatly enhanced by the development of this alloy. The material was added to the ladle just before pouring, and analyses of steels made during this period indicated that the standard practice was to add 0.15 to 0.25 percent manganese in this manner. The beneficial effects of this addition saved as much as 40 percent of the material usually discarded (106).

The influence of manganese on the quality of the steel was first attributed to the reduction of dissolved oxides in liquid metal. Henderson, however, pointed out that the alloyed manganese might improve the physical properties of the iron and furthermore might react with the sulphur in the steel. He suggested investigation of these two effects. Publications in this connection, which appeared soon after his suggestion, indicated that his contentions were somewhat justified.

The effects of high percentages of manganese on the properties of iron were not discovered until 1882, when Hadfield published and patented the results of his investigations. Steels were made containing as much as 10 to 15 percent manganese, and the properties of these materials will be discussed in detail later. The following pages will also include a summary of information on the reactions of manganese in liquid steel and the effects of manganese on the properties of the finished material.

MANGANESE IN LIQUID-STEEL METALLURGY

Sources of Manganese Used in
Steel Making

During the manufacture of steel manganese is introduced in various ways. Nearly all present-day methods of manufacture involve the utilization of steel scrap, which contains variable amounts of manganese. In some processes, such as the arc electric furnace, where steel is usually made entirely from scrap, the manganese content of the finished material depends only upon the manganese content of the scrap charged and final additions of manganese-containing ferro-alloys, such as spiegel, ferromanganese, etc. In such processes as the acid and basic open hearths, however, the manganese content of the finished material also depends upon its concentration in the pig iron charged.

Presence of Manganese in Steel

Manganese may be present in steel in several different forms. As will be shown later, manganese is entirely soluble in liquid iron and quite appreciably soluble in solid iron. The largest portion of this material exists in solid or liquid solution as an alloy; however, other forms are present, but to a much lesser degree. Some of the manganese exists as manganese carbide (Mn_3C), probably in conjunction with the iron carbide of similar composition. Other forms in which manganese may exist in steel are such compounds as manganous oxide and manganous sulphide, products of the reactions of manganese with iron oxides and iron sulphides, respectively. The phenomena that accompany the presence of these various forms of manganese in steel will be discussed in the following pages.

Ferrous Alloys of Manganese -- Phase Diagrams

Nearly all ferrous alloys of manganese exist in the quaternary system iron-manganese-carbon-silicon, and most of these have compositions in the close proximity of the iron-manganese binary system.

High- and low-carbon ferromanganese and the silicomanganese alloys have compositions near the manganese-rich side of this binary, whereas low- and high-carbon spiegels (and silicospiegels) and the manganese steels are all located near the iron-rich side of the iron-manganese system.

As has been pointed out in various sections of this paper, the manganese-rich alloys are used chiefly as addition agents (deoxidizers, etc.) in steel manufacture. Our main interest in these materials therefore lies in the ease and economy with which they can be manufactured and used and not necessarily in their physical properties as alloys.

On the other hand, iron-rich manganese alloys are extremely interesting from the standpoint of their physical characteristics, and one of the following sections is devoted to the properties of the manganese steels.

In the following pages the available data concerning the six binary and the four ternary phase diagrams constituting the basis for the quaternary system iron-manganese-carbon-silicon are discussed.

Binary Systems

System Iron-Manganese. - The system iron-manganese, as determined by various investigators, is shown in figure 35. The liquidus and solidus are given according to Rumelin and Fick (330) and Levin and Tammann (233), whose values are all in fair agreement. These curves indicate that at high temperatures iron and manganese form continuous series of solid and liquid solutions. The only apparent interruption occurs in the iron-rich side of the diagram and is due to the delta-gamma iron transformation. This portion of the diagram was established by Rumelin and Fick. They found that the melting point of pure iron was $1,528^{\circ}\text{C.}$, and its delta-gamma transformation occurred at $1,401^{\circ}\text{C.}$ This transformation temperature is raised by small manganese additions, and between 2 and 10 percent manganese the transformation begins at $1,454 \pm 2^{\circ}\text{C.}$ Above 10 percent manganese the delta-gamma transformation cannot be detected by thermal analysis, and it is believed that alloys containing more than 12 percent manganese solidify with the formation of the gamma solid solution.

Iron-rich alloys. - The effect of manganese on the gamma-alpha transformation in the iron-rich alloys is shown graphically in figure 35. With increasing manganese contents this transformation, which occurs in pure iron at 900°C. , is lowered rapidly. This side of the diagram has been studied by various investigators, chiefly Ohman (282), Walters (373), Schmidt (339), Dejean (64), Bain (12), and Ishiwara (177).

In general, all their values are similar. Walters and Ohman probably had the purest alloys and checked their findings by various experimental methods.

The results of these investigations indicate that the lowering of the gamma-alpha iron transformation is characterized by the area ABC in the figure. Above and to the right of this area the face-centered gamma solid solution is the stable phase, whereas below and to the left of this region the body-centered alpha solid solution is stable. Alloys having compositions within this area contain both phases. This immiscibility region is limited at room temperatures by the values obtained by various workers. Point C was established by the work of Ohman (282), Ishiwara (177), Schmidt (339), and Bain (12), whereas point B was established by Bain.

Schmidt's X-ray investigations introduced a rather interesting complication to this portion of the diagram. He found that alloys containing 12 to 29 percent manganese (or within the above mentioned immiscibility range) exhibit a hexagonal close-packed structure. This he called the epsilon phase. Although several other investigators (Ohman, Ishiwara, and Walters) have also identified this phase, there is considerable uncertainty as to its significance. Schmidt suggests that it is a solid solution of iron and manganese which is stable only at low temperatures. Ohman states that this

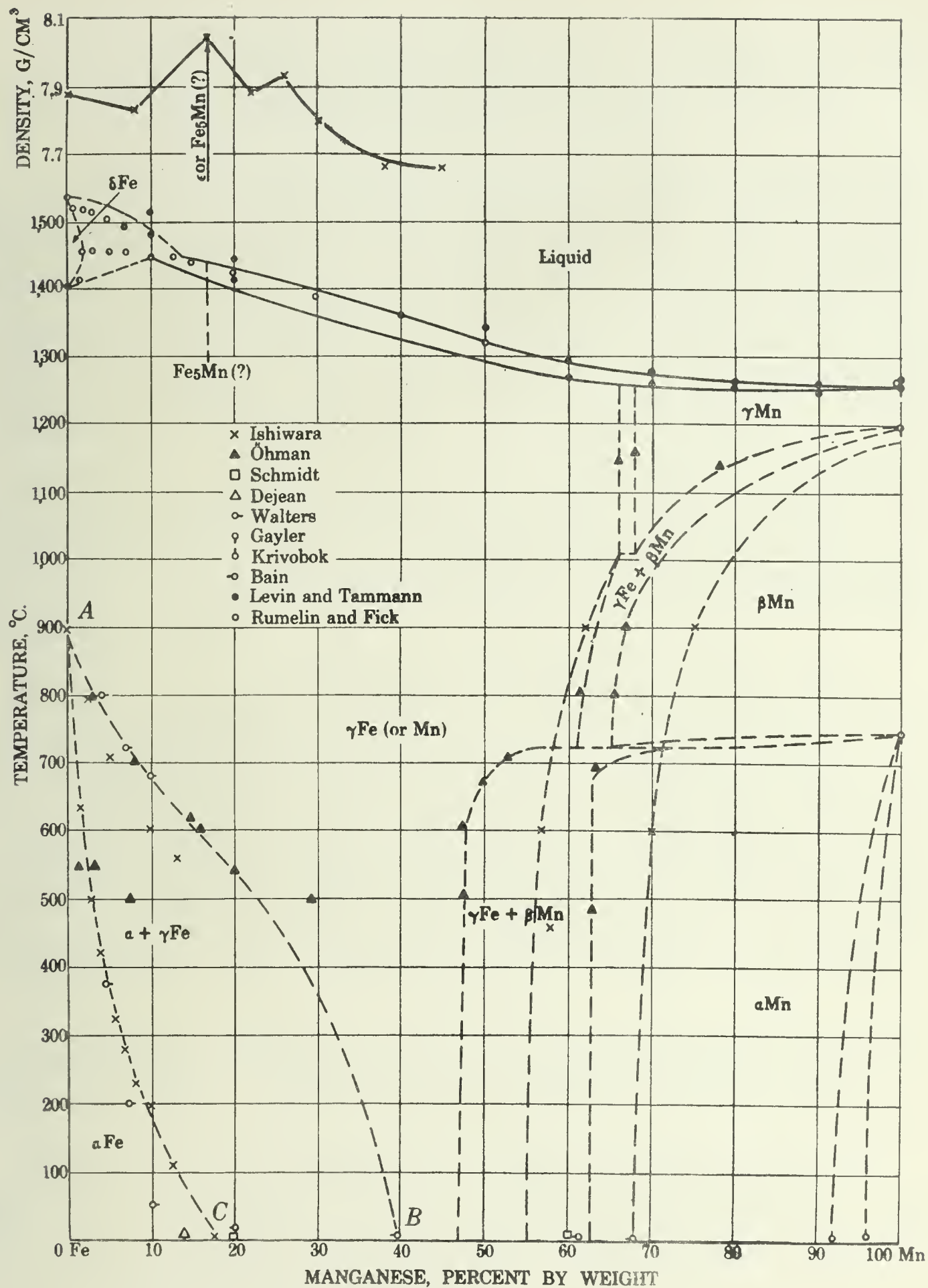


Figure 35.- System iron-manganese.

phase is not stable over 500°C . and is formed only by the decomposition of the gamma-iron phase and the simultaneous formation of a supersaturated alpha-iron phase. This was termed alpha-prime instead of epsilon (according to Schmidt), and the significance of this term is apparent.

However, the almost simultaneous publication by Ishiwara suggested that the origin of this new phase (epsilon or alpha-prime) was the compound Fe_5Mn , which according to him is stable only at high temperatures. There is little confirmatory evidence in favor of this theory other than his density measurements shown at the top of figure 41. However, Ohman states that the epsilon phase is only stable when rapidly cooled and that the strongest indication of the hexagonal epsilon phase was found in an alloy containing 22.9 atomic percent of manganese and quenched at $1,100^{\circ}\text{C}$. The atomic percentage of manganese in Fe_5Mn is 16.66 percent, and the above alloy would contain a large portion of this compound, provided of course that it exists. This question will probably be investigated further by some authors already mentioned herein.

Manganese-rich alloys. - The properties of pure manganese have already been discussed in chapter 1 and will not be repeated. However, for the purposes of this section it should be recalled that manganese exhibits two allotropic transformations (107). Cubic alpha manganese is stable from 0° to 742°C ., whereas cubic beta manganese is stable from 742° to $1,191^{\circ}\text{C}$. From the latter temperature up to the melting point ($1,245^{\circ}\text{C}$.) the face-centered tetragonal form (gamma) is stable.

The effect of iron on the manganese transformation has been studied. There is some disagreement in the results of the various investigations, although the general trends are similar. As shown in figure 35, iron rapidly lowers the gamma-beta transformation from $1,191^{\circ}\text{C}$. at 0 percent iron to 725°C . at approximately 40 percent iron. Some investigators, such as Ishiwara (177) and Krivobok (219), concluded that the gamma-beta transformation was lowered to room temperature at approximately 40 percent iron. Ohman's investigations, however, indicated that the effect of the beta-alpha manganese transformation was quite apparent. With increasing iron content this transformation was lowered from 742° to 725°C . at 60 percent iron. Hence an alloy containing more than 40 percent iron will transform directly from the gamma (iron or manganese) solid solution into the alpha without passing through the intermediate form of beta solid solutions.

The gamma-beta immiscibility ranges are shown in the figure according to Ohman and Ishiwara. Because of the purity of Ohman's samples his values are probably more correct. Ishiwara plotted the amount of "Manganese added" rather than analytical values, and his conclusions are somewhat questionable.

Ohman also gives the immiscibility ranges of the gamma-alpha manganese solid solutions as ranging from 48 to 63 percent manganese at 500°C . and assumes that nearly the same limits are maintained at room temperature. Bain's limits for this immiscibility region ranged from 61 to 68 percent manganese at room temperature, whereas Schmidt found that the lower manganese limit was at 60 percent manganese at room temperature.

Krivobok stated that according to his metallographic studies the beta-manganese solid solution was stable at room temperature between 65 and 92 percent manganese. Also, gamma and beta manganese were immiscible between 58 and 65 percent manganese, and similarly the beta and alpha immiscibility range varied from 92 to 96 percent manganese at room temperature. Hence alpha manganese was only stable from 96 to 100 percent manganese or conversely 0 to 4 percent iron. These results have not been substantiated by X-ray studies. According to the above abstracts iron and manganese form complete series of solid and liquid solutions at high temperatures, but due to the various allotropic transformations in both iron and manganese such is not the case at room temperatures.

The X-ray studies of Bain, Schmidt, Ohman, and Ishivara on iron-manganese alloys at room temperatures have been summarized briefly in the following table:

TABLE 84. - Composition Ranges of Stable Solid Solutions in the Iron-Manganese System

Solid solution	Concentration limits at 25° C., percent manganese	Crystal structure
Alpha iron	0-18	Body-centered cubic.
Alpha + gamma iron ¹	18-39	Body-centered and face-centered cubic.
Epsilon (or Fe ₅ Mn?)	13-29	Hexagonal close-packed.
Gamma iron	39-47 (or 55)	Face-centered cubic.
Gamma + alpha manganese ¹	47 (or 55) to 63 (or 68)	Face-centered cubic and cubic.
Alpha manganese	63 (or 68) to 100	Cubic 56 atoms per unit cell

¹ Two immiscible solutions.

Osawa (235) and Ohman (232) studied the effects of increasing manganese contents on the crystal structure of iron and the effects of iron on the space-lattice constants of alpha, beta, and gamma manganese.

Benedicks (21) studied the effect of manganese on the density of liquid iron at 1,600° C. Only a very slight decrease was noted up to 19 percent manganese.

System Manganese-Carbon. - This is essentially the manganese-manganese carbide (Mn₇C) system, and up to the composition of Mn₇C these two materials are completely miscible in the liquid state. This has been determined by Stadler (353) whose phase diagram is shown in figure 36. Unfortunately, Stadler's temperature for the melting point of manganese is some 45° C. lower

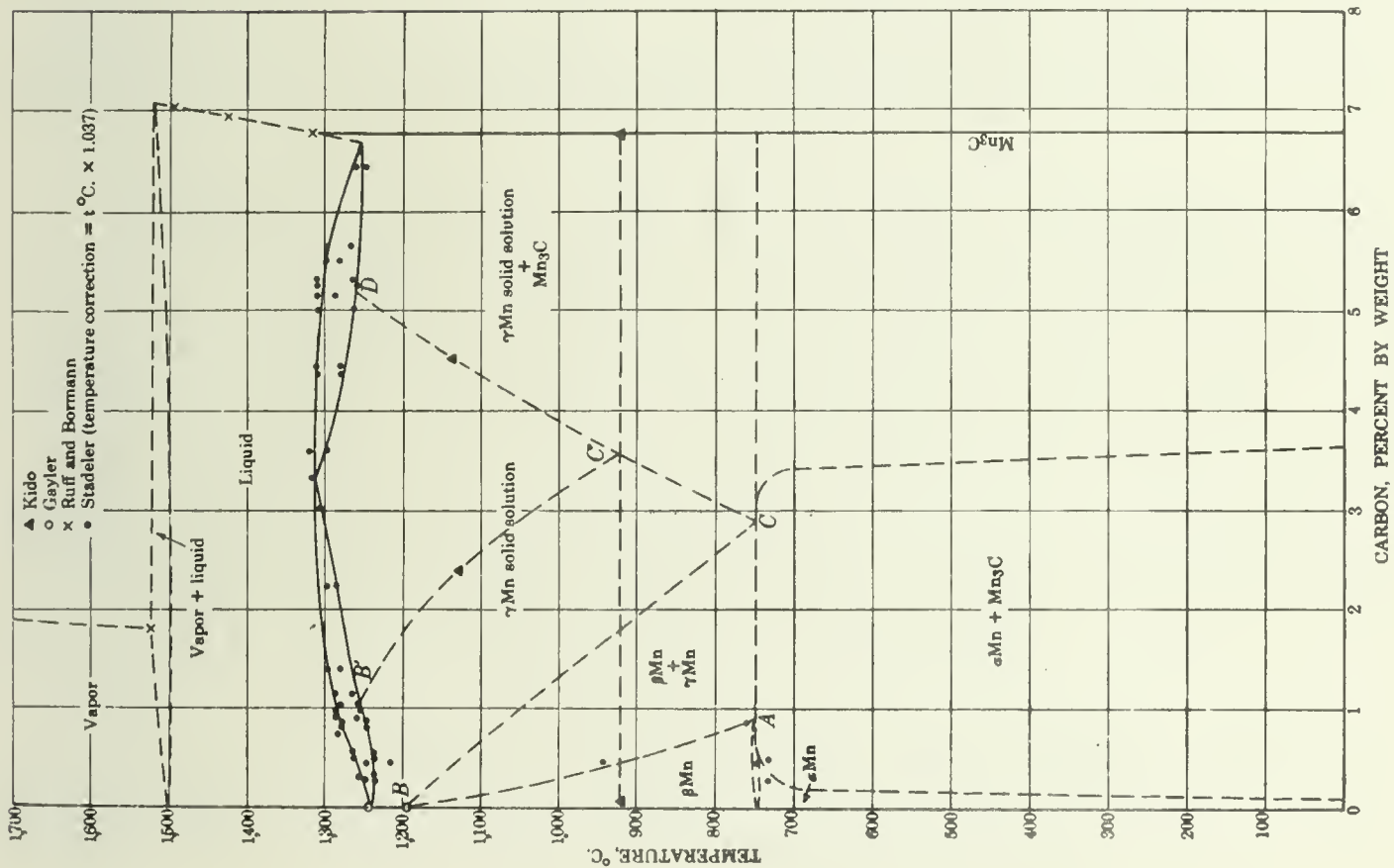


Figure 36. - System manganese-carbon.

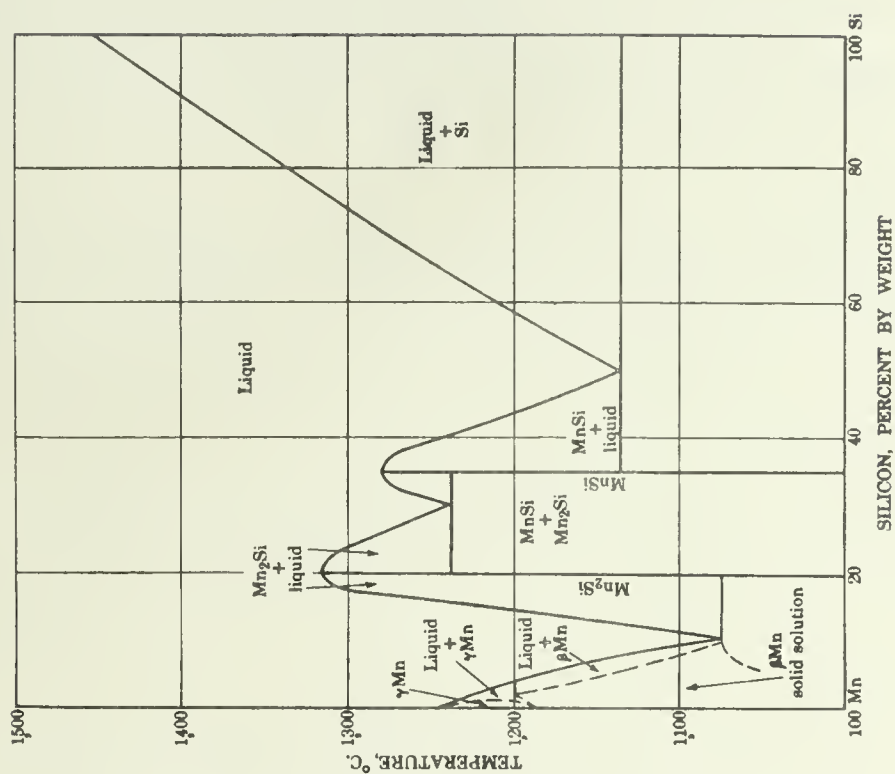


Figure 37. - System manganese-silicon, according to Dörinckel.

than Gayler's (107), and the points plotted in figure 36 have been corrected proportionally for this amount by multiplying by $\frac{1.245}{1.200} = 1.0375$. These plotted values show a liquidus having a maximum temperature ($1,315^{\circ}\text{C.}$) at about 50 percent Mn_3C (or 3.4 percent carbon). The temperature drops gradually on either side of this point to the melting points of manganese and Mn_3C .

Ruff and Bormann (328) determined the solubility of carbon in liquid manganese at temperatures above the melting point of liquid Mn_3C . Their values, which are also shown in figure 36, accord with Stadeler's corrected values. Ruff and Bormann also determined the boiling points of manganese-carbon alloys, and these data are represented diagrammatically in figure 36.

The characteristics of the solid alloys are not so well-established as in the iron-manganese system. Stadeler determined the limits of the solid solubility of Mn_3C in manganese and conversely manganese in Mn_3C , and stated that solid alloys bounded by the points 0.1 percent C, A, C, and 3.6 percent C are composed of two immiscible solid solutions, one of which is manganese-rich whereas the other is carbide-rich.

Kido (202) studied Mn- Mn_3C alloys by thermal means and concluded that the system contains a eutectoid (C in figure 36). According to Kido the region B'C'D of figure 36 contains the homogeneous solid solution of gamma manganese in Mn_3C . Alloys left of line B'C' contain two phases, gamma manganese and the above solid solution. Alloys right of line C'D are composed of two phases, the solid solution and the carbide.

The eutectoid temperature (920°C.) was supposed to be approximately the same as the beta-alpha manganese transformation. This however is not the case, as was shown previously. This transformation takes place at 742°C. If we assume that there is a eutectoid in the system and correlate this assumption with Stadeler's results, it is possible to construct a much more reasonable diagram, such as that indicated by points A, B, C, and D.

This system extends the gamma-manganese solid solution over to the gamma-beta transformation in pure manganese at point B. The eutectoid temperature and composition represented by point C agree very well with Stadeler's immiscibility range described above. Likewise his values agree favorably with point A, which has been taken as the limiting solubility of Mn_3C in beta manganese at the beta-alpha transformation.

Other points of Stadeler's that agree with this diagram are the following: 4.5 percent C, 940°C. ; 12.5 percent C, 960°C. ; and 3.6 percent C, 887°C. , all of which are shown in the figure. Because of the uncertainty of the solid solubility limits all lines have been dotted in the figure.

At present these alloys have but little practical use, but the system Mn- Mn_3C has great theoretical interest because it confirms the existence of Mn_3C -- an item of great importance to ferrous metallographists.

Ferromanganese is the only practical alloy having compositions approximately within this system. High-carbon ferromanganese is essentially a mixture of iron and manganese carbides ($Mn_3C + Fe_3C$), inasmuch as it contains about 7.5 percent carbon, 80 percent manganese, and 12.5 percent iron. There are probably only infinitesimal amounts of uncombined iron or manganese present. Low-carbon ferromanganese, containing 2 to 3 percent carbon, 80 percent manganese, and 17 to 18 percent of iron, probably contains a large percentage of the eutectoid represented by point C, figure 36.

System Manganese-Silicon. - According to the specific-gravity determinations of Frilley (100), four silicides of manganese should exist -- Mn_2Si , $MnSi_2$, $MnSi$, and Mn_5Si . However, other researches have identified only two -- Mn_2Si and $MnSi$. The existence of these two silicides has been confirmed by the melting-point work of Dörinckel (74), whose results are shown diagrammatically in figure 37. The only change in his values which has been made in constructing this diagram is the melting point of silicon. This has been plotted at $1,458^\circ C.$, according to the selected value given by Mellor (252).

No account has been made by Dörinckel of the gamma-beta manganese transformation, and for this reason the tentative break in the solidus of the high-manganese alloys has been made. This is given in a manner similar to the delta-gamma iron transformation shown in the system iron-manganese fig. 35).

Little is known about the physical properties of these alloys at room temperatures. In fact, the analyses of few if any commercial alloys are to be found strictly within this binary system. The more practical alloys at present are to be found within the quaternary system iron-manganese-silicon-carbon.

Other Binary Diagrams in Quaternary System Iron-Manganese-Carbon-Silicon. - The following manganese-free binary systems are given briefly herein because of their importance in connection with manganese alloys.

System iron-carbon. - This system is so well known that it will not be discussed in detail herein. The equilibrium diagram is reproduced in figure 38 according to the selected values by Ralston (310). This figure might more appropriately be called the system $Fe-Fe_3C$. A recent publication by Körber and Olsen (212) is particularly interesting, inasmuch as they have represented in one diagram nearly all of the experimental data pertaining to this system. As shown in the figure, this system includes one eutectic and one eutectoid as contrasted with the $Mn-Mn_3C$ system, the liquidus of which exhibits a maximum with no compounds or eutectics. As stated in the above discussion, however, the latter system may contain a eutectoid. The connection between this and the $Mn-Mn_3C$ system will be discussed in connection with the ternary diagram iron-manganese-carbon.

System iron-silicon. - The system iron-silicon, as shown in figure 39, was constructed according to the diagrams given by Mellor (253) and the International Critical Tables (176). Except for a few discrepancies in temperatures the two systems described by these authorities are the same up to about 40 percent silicon. The $FeSi_2$ solid-solution (epsilon) range was

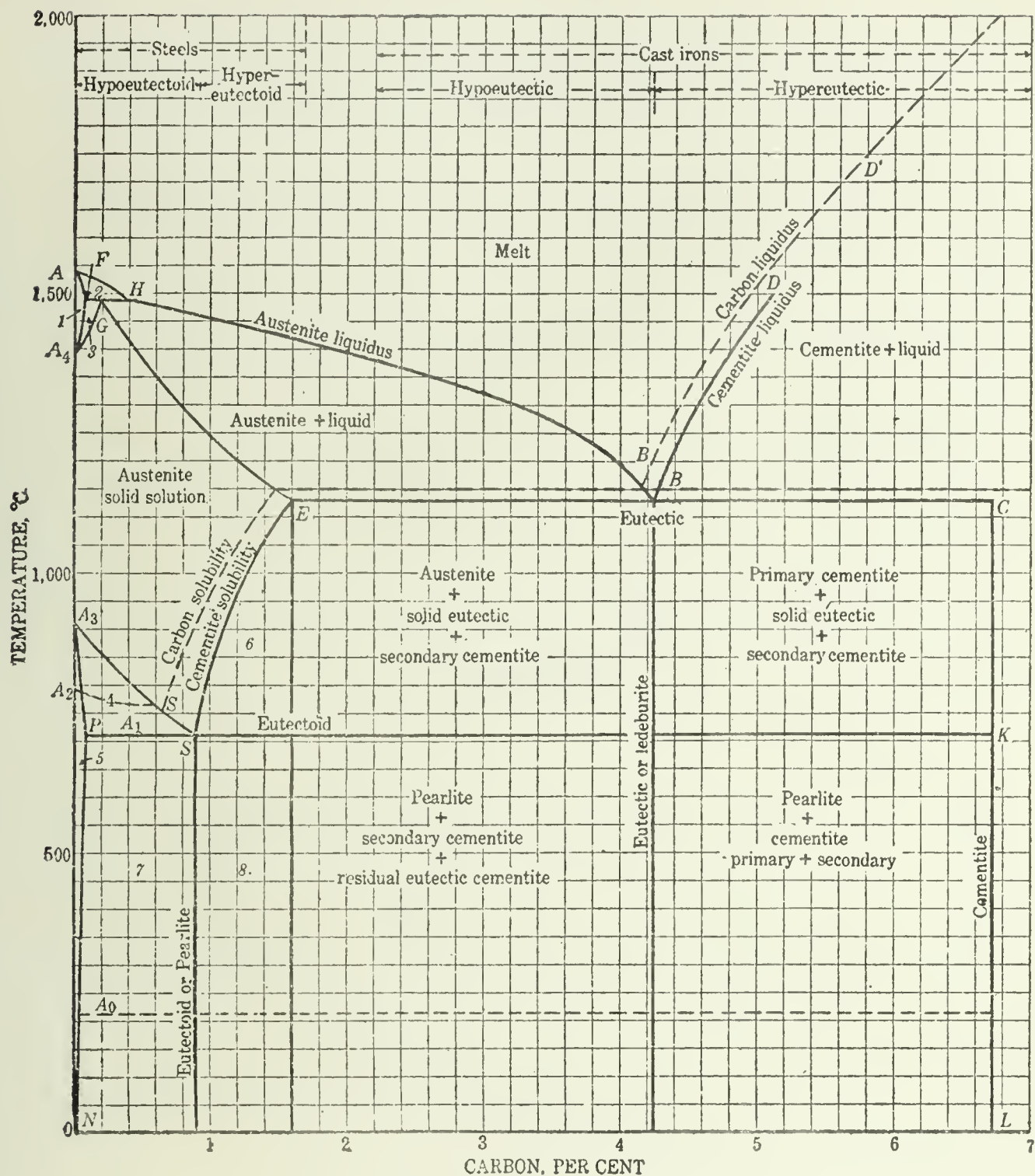


FIGURE 38.—Iron-carbon temperature-composition phase diagram: Area 1, δ ferrite, maximum solubility of carbon 0.07 per cent; area 2, δ ferrite + melt; area 3, δ ferrite + austenite; area 4, α ferrite + austenite; area 5, α ferrite, maximum solubility of carbon 0.034 per cent; area 6, austenite + proeutectoid cementite; area 7, proeutectoid α ferrite + pearlite; area 8, proeutectoid cementite + pearlite. Point A_0 , temperature of magnetic transformation of Fe_3C , 210° C.; point A_1 , temperature of eutectoid (splitting of solid austenite into α ferrite and cementite), 720° C.; point A_2 , temperature of magnetic transformation of α ferrite, 795° C.; point A_3 , temperature of transformation of γ ferrite to α ferrite, 906° C.; point A_4 , temperature of transformation of δ ferrite to γ ferrite, 1,400° C.; point A , melting point of pure iron, 1,535° C.; point N , solubility of carbon (as cementite) at room temperature, 0.003 per cent C.

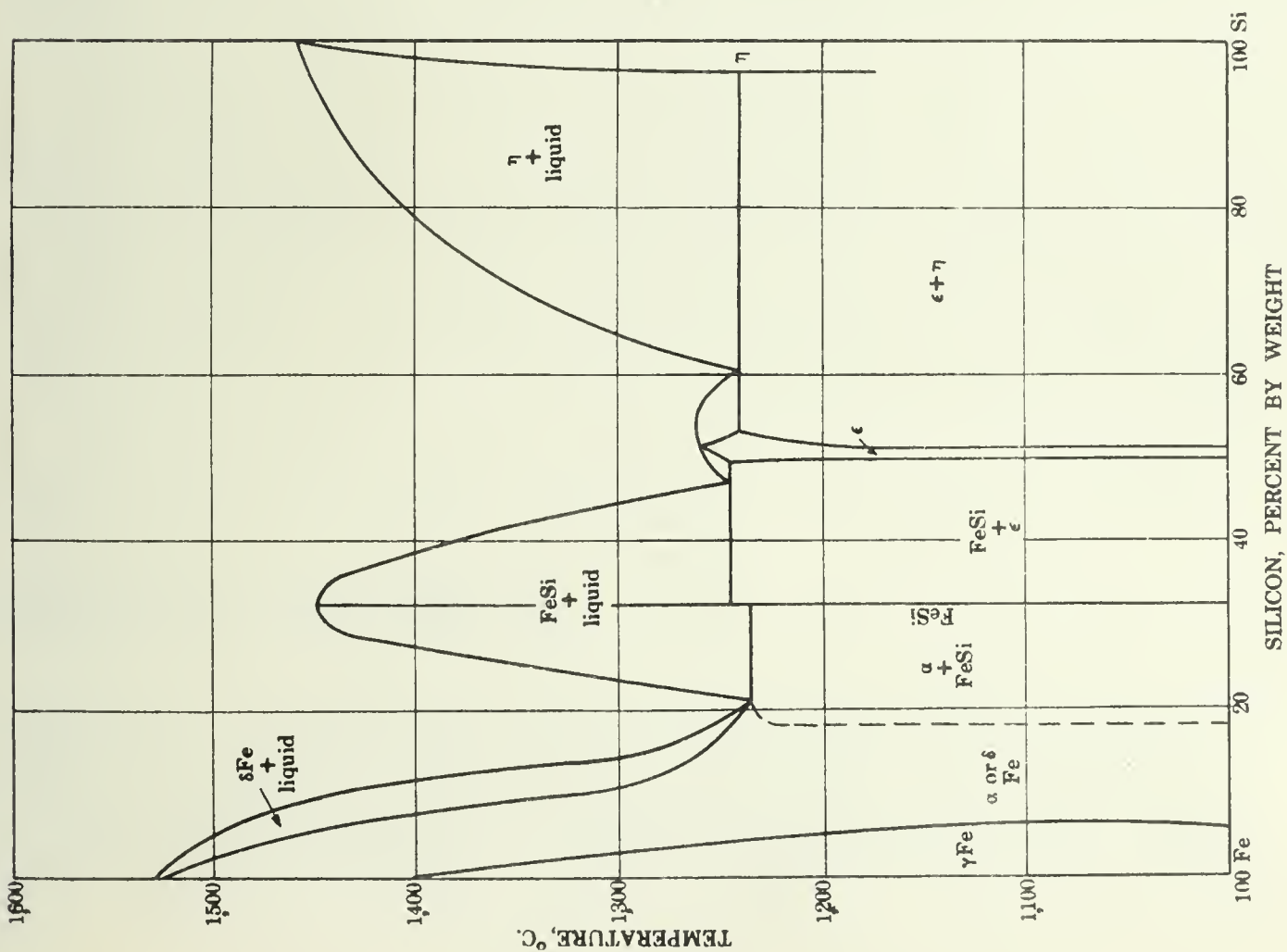


Figure 20.—System iron-silicon.

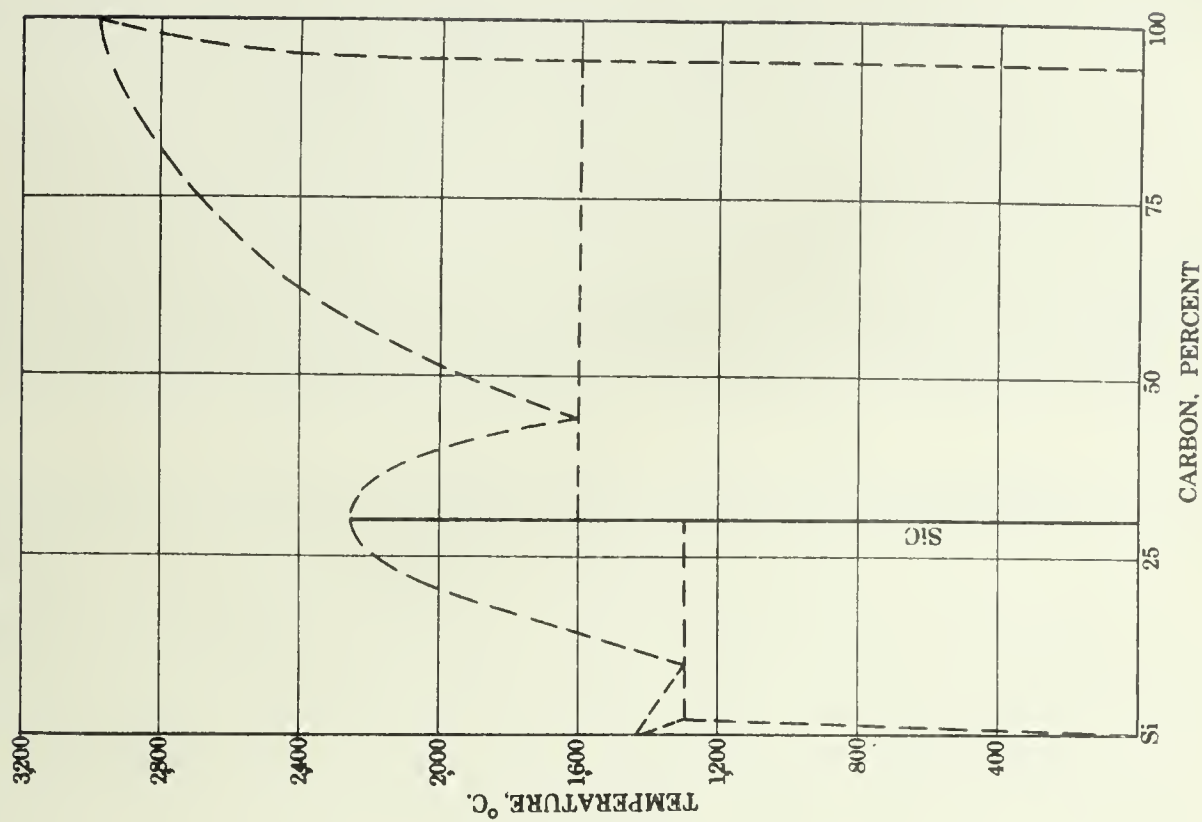


Figure 40.—Tentative system silicon-carbon.

not given by Mellor; likewise the range of the γ -solid solution was not considered by this author. The melting point of silicon was given according to Mellor's previously mentioned "selected" value -- $1,458^{\circ}\text{C}$.

The discrepancies between these two publications are probably due to the literature to which they referred. Neither referred to work cited by the other.

The data considered by Mellor were obtained by Murakami (268), Gontermann (119), Lowzow (235), Kurnakoff and Urasoff (223), and Guertler and Tammann (127). The diagram given by the International Critical Tables was constructed according to the works of Gruneisen (125), Herz (160), Johnston and Adams (181), and Cohen and Helderman (54).

The correlation of this system with the system iron-silicon-manganese will involve discussions of greater interest for the purposes of this paper than a complete discussion of this binary, at this time.

System silicon-carbon. - Because of the extremely high temperatures involved very little is known about this system. There is little question, however, as to the existence of the compound SiC (carborundum), which was first discovered by Schützenberger (344). This compound has a high melting point ($2,227^{\circ}\text{C}$), according to the estimates of Weigel (382). The existence of two other compounds has been suggested. Silicon dicarbide (SiC_2) was supposed to have been prepared by Colson (56), whereas Kohn-Abrest (251) believed he had prepared C_5Si_3 . The existence of these compounds has not been checked by subsequent investigations. These two latter investigators may have obtained a eutectic in the system silicon-carbon, which was composed of SiC and C.

If such were the case this system would have the appearance of the very tentative diagram in figure 40. The melting points of silicon, SiC, and carbon are all known with a fair degree of accuracy (that is, $1,458^{\circ}$, $2,227^{\circ}$, and $3,000^{\circ}\text{C}$. respectively), and these values have been used in constructing this tentative diagram. The compositions of the two compounds SiC_2 and Si_3C_5 are 46.2 and 41.6 percent carbon, respectively. By assuming that the discoverers of these two compounds really were dealing with a eutectic, the eutectic composition was taken as the mean of these two compositions. The eutectic temperatures have both been assumed in constructing this diagram.

The above assumptions have been made for the purpose of studying the ternary and quaternary systems involving this binary. This is the only binary that is not known with fair accuracy.

Ternary-Phase Diagrams in Quaternary System Iron-Manganese-Silicon-Carbon

There are four possible ternary diagrams in the quaternary system under consideration -- the systems iron-manganese-carbon, iron-silicon-carbon, iron-manganese-silicon, and manganese-silicon-carbon. Unfortunately, no information was found applicable to the latter two systems. Furthermore, only small areas in the first two systems have been investigated. Certain attempts have been made to extrapolate the influence of the various binary

systems into the ternary, but these studies will not be given herein. Such extrapolation has most often led to erroneous conclusions.

The determination of the liquid- and solid-phase equilibria in all four systems should represent an excellent field for fundamental research.

The effects of manganese and silicon on the solubility of carbon in iron have been determined and will now be discussed under the headings "The System Iron-Manganese-Carbon" and "The System Iron-Silicon-Carbon." The data are too meager, however, to establish these systems completely.

System Iron-Manganese-Carbon. - Manganese increases the solubility of carbon in iron, according to Wüst (398). His data show that a sample containing 0.06 percent manganese can hold only 4.28 percent carbon in solution. With 4.23 percent manganese, 4.97 percent carbon can be present, whereas with higher percentages of manganese -- 12.66, 59.73, and 80.45 -- the melt can contain 5.38, 6.76, and 6.90 percent carbon, respectively.

The freezing points of nearly all Wüst's samples were virtually at the same temperatures ($1,120^{\circ}$ to $1,146^{\circ}$ C.) in most instances. Probably these temperatures really represent the freezing point of a ternary eutectic. If this is true Wüst merely determined the temperature at which the last portions of his melts solidified on cooling, or the first portions became liquid on heating. The above-mentioned carbon solubilities are therefore indeterminate because the fourth variable, temperature, was unknown.

The tendency for manganese to increase the solubility of carbon in liquid iron was also observed by Nordbury (276), Goerens (115), and Kürten (225). The latter author stated that ferromanganese can contain as much as 7.5 percent of carbon, an observation often made with regard to this material.

Oberhoffer (278) reviewed the work of Lütke and concluded that ternary alloys included in an area formed by drawing lines between the points manganese, 0 percent, carbon, 1.8 percent; manganese, 35 percent, carbon, 3.5 percent; and manganese, 0 percent, carbon, 6.67 percent contained two heterogeneous phases at room temperature. One phase was iron-rich, whereas the other was carbon (or carbide) rich; also the alloys within this range of composition were not malleable.

System iron-silicon-carbon. - Wüst (398) observed that when silicon is added to low-manganese pig iron carbon is precipitated from the liquid. The eutectic composition (in the system iron-carbon) decreased from 4.3 percent with low silicon to about 0.87 percent when 27 percent silicon was added. Other investigators who observed this effect in liquid melts were Charpy and Cornu (46), Andrew (8), and Honda and Murakami (166).

Gontermann (119) determined the primary crystallization temperatures of low-silicon alloys (up to 10 percent silicon). Isotherms plotted from his experimental data clearly indicate the above-mentioned carbon-solubility-reducing tendency of silicon. The solidus in nearly every sample was in the same temperature range ($1,076^{\circ}$ - $1,168^{\circ}$) and most of these

temperatures ranged only from $1,121^{\circ}$ to $1,150^{\circ}$, an average of $1,135^{\circ}$ C. This is probably the freezing point of a ternary eutectic.

Various investigators studied the effect of silicon on the solubility of carbon in solid iron. Chief among these were Charpy and Cornu (47), Becker (18), Hanson (146), Pearson (299), Yensen (402), and Kriz and Poboril (221). Their data are not adequate, however, for constructing the ternary system.

Quaternary System Iron-Manganese-Silicon-Carbon

In spite of the greater number of variables in this system, considerable information has recently been made available through the investigations of Herty and Royer (157). It is rather difficult to vision a four-component system, but the picture can be simplified in this particular instance by stating that the above authors determined the solubility of carbon in iron-manganese-silicon alloys at specified temperatures. This may be clarified further by reference to their diagram reproduced in figure 41. The base of this figure is the ternary system iron-manganese-silicon, whereas the vertical axis represents the carbon concentration in the saturated alloy. The curved surface represents the saturated-carbon content of all alloys at $1,700^{\circ}$ C., and its limits are represented as follows:

TABLE 85. - Limiting Carbon Contents in
System Fe-Mn-C-Si

Alloy:	Percent C, Saturated
100 percent Mn	7.84
100 percent Fe	5.90
42 percent Mn, 58 percent Si	.00
46 percent Fe, 54 percent Si	.00

A similar but lower (concentric) curved surface would be obtained by graphic representation of the carbon saturation values at lower temperatures ($1,300^{\circ}$ and $1,500^{\circ}$ C.).

It is obvious from figure 41 and table 85 that manganese overcomes the carbon-precipitation tendencies of silicon more than iron. For example, 58 percent of silicon is required to decrease the carbon solubility to zero in a manganese-silicon alloy, whereas 54 percent is required to prevent carbon solubility in an iron-silicon alloy. The greater solubility of carbon in manganese than in iron is also shown in the table and figure, manganese dissolving some 2 percent more carbon than iron at $1,700^{\circ}$ C. Extrapolation of the solubility curve to 100 percent manganese was necessary, but an experimental iron-manganese melt containing 80 percent manganese was saturated with 7.14 percent carbon at $1,500^{\circ}$ C. Iron is saturated with 5.40 percent carbon at the same temperature.

The graphitizing action of silicon was verified in the above experiments and is indicated by the insolubility of carbon in high-silicon, ternary alloys. The insolubility range varies from iron-silicon alloys containing more than 54 percent of silicon, to manganese-silicon alloys containing more than 53 percent of silicon.

The practical significance of this work may best be illustrated by figure 42, wherein the triangular base of the quaternary system is represented. The limiting iron, manganese, and silicon contents of the present commercial alloys have been represented without respect to their carbon contents.

The manganese steels are represented as being on the iron-manganese binary line. Actually these materials contain less than 0.2 to 0.3 percent silicon. These materials are not so important with regard to carbon saturation, because care is taken to maintain the unsaturated condition. The carbon content of these steels is seldom greater than 1.0 percent, whereas the saturation values represented in figure 41 range from about 5.4 to about 6.3 at 1,500° C.

The analyses of ordinary spiegel and ferromanganese are represented in areas adjacent to the iron-manganese binary because these materials usually contain 1 to 3 percent silicon. The carbon-saturation values for these materials are important and may be estimated by reference to figure 41.

The special steel-deoxidizing alloys (the silicospiegels and silico-manganese) such as those developed recently by Herty and Fitterer (152) are represented according to their analyses in figure 42. The carbon-saturation values for these materials may be estimated by reference to figure 43, which also has been reproduced from Herty and Royer's work. These concentrations were determined by intersecting the solid figure 41 by two planes intersecting the base (shown in fig. 42) at lines AB and AC.

Unless it is otherwise specified, spiegel, silicospiegel, silicomanganese, and ferromanganese are usually saturated with carbon at the manufacturing temperature in accordance with these figures. Occasionally a low-carbon alloy is desirable for addition to low-carbon steel, and unsaturated alloys are requested. Low-carbon ferromanganese, for example, contains only 1 to 2 percent carbon. When such materials are added to low-carbon steels it is not so difficult to meet the manganese and carbon specifications for the heat.

The saturation of carbon in the ferrosilicons can also be estimated according to their limiting analyses plotted in figure 42 and the carbon saturation in figure 41. It will be readily seen that the carbon solubility in such materials is quite low.

The diamond at the top of figure 42 represents the analyses (except for carbon) of the ordinary pig and cast irons. A more complete representation of this diamond on a greatly magnified scale would be very interesting, but it is not important for the purposes of this paper.

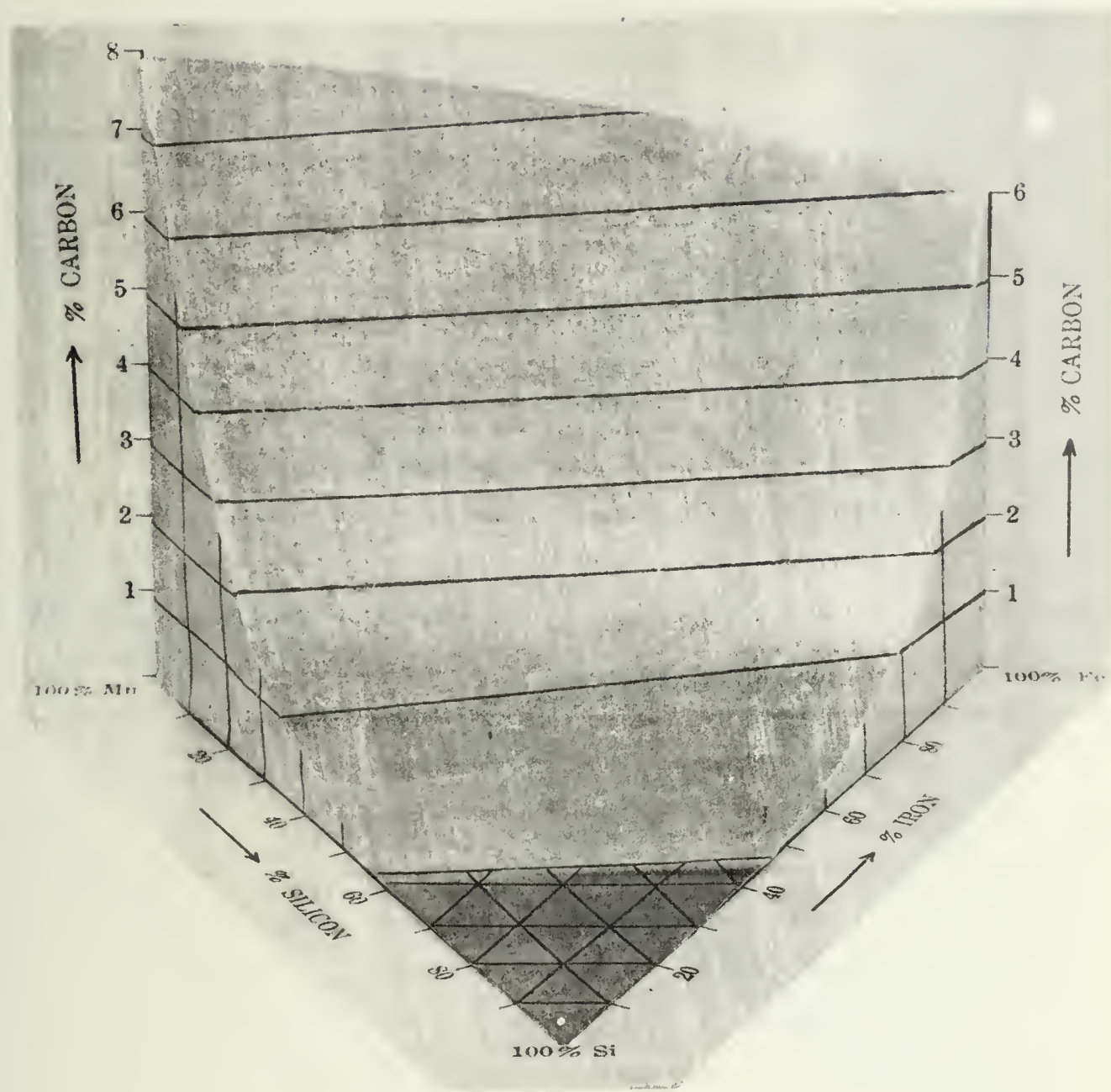


Figure 41.- Carbon solubility at 1,700 °C. in Fe, Mn, and Si alloys.

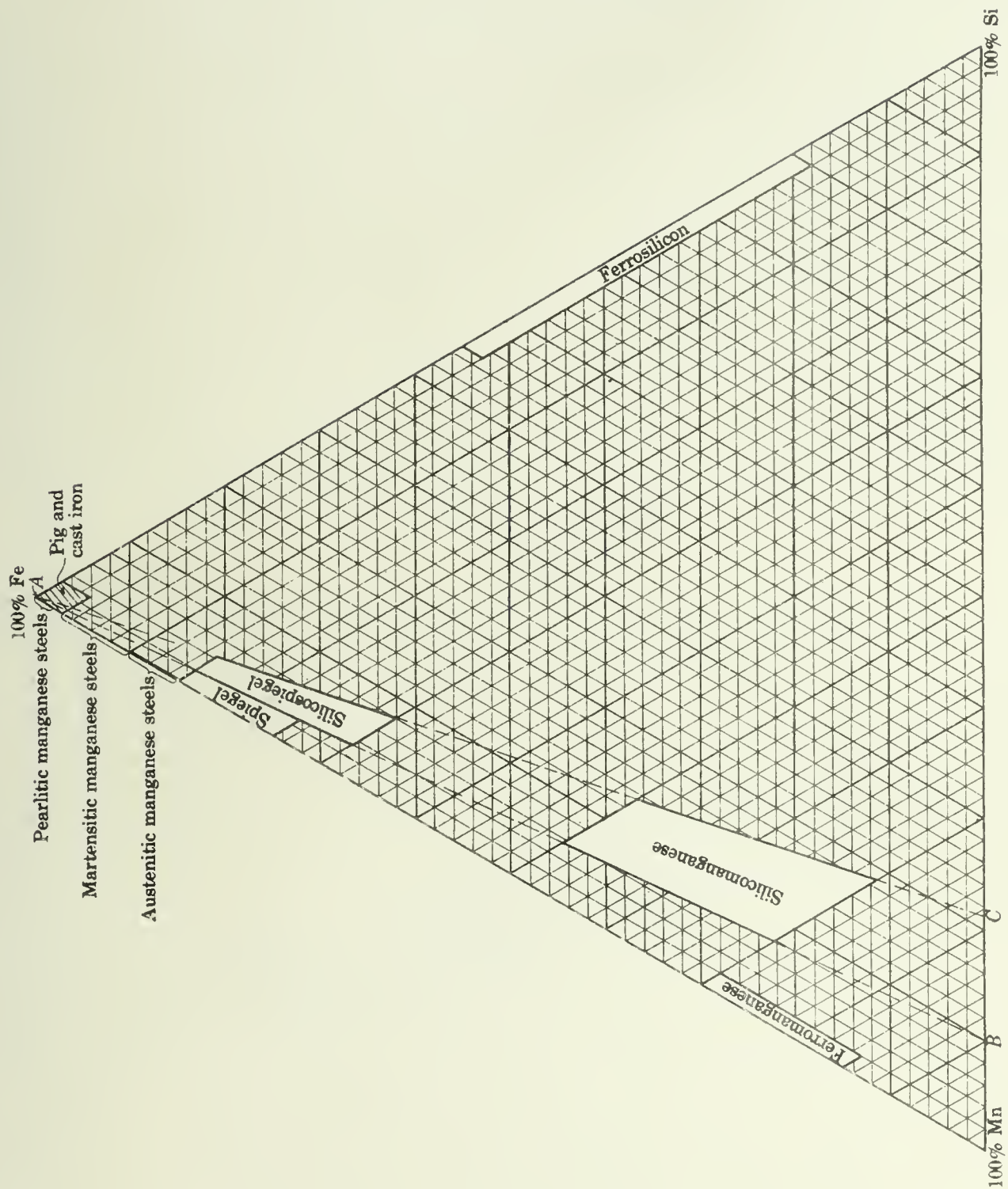


Figure 42.- System iron-manganese-silicon.

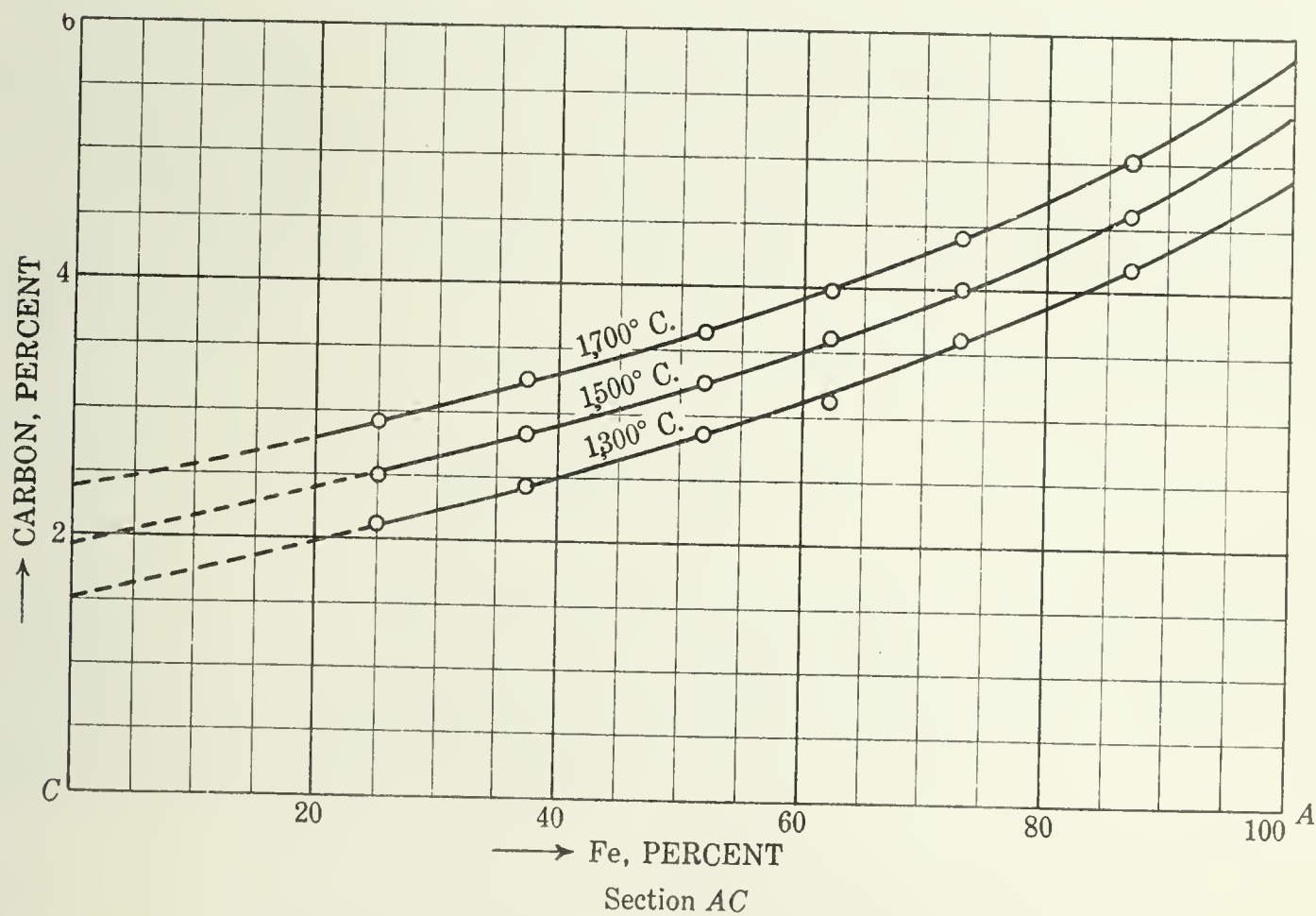
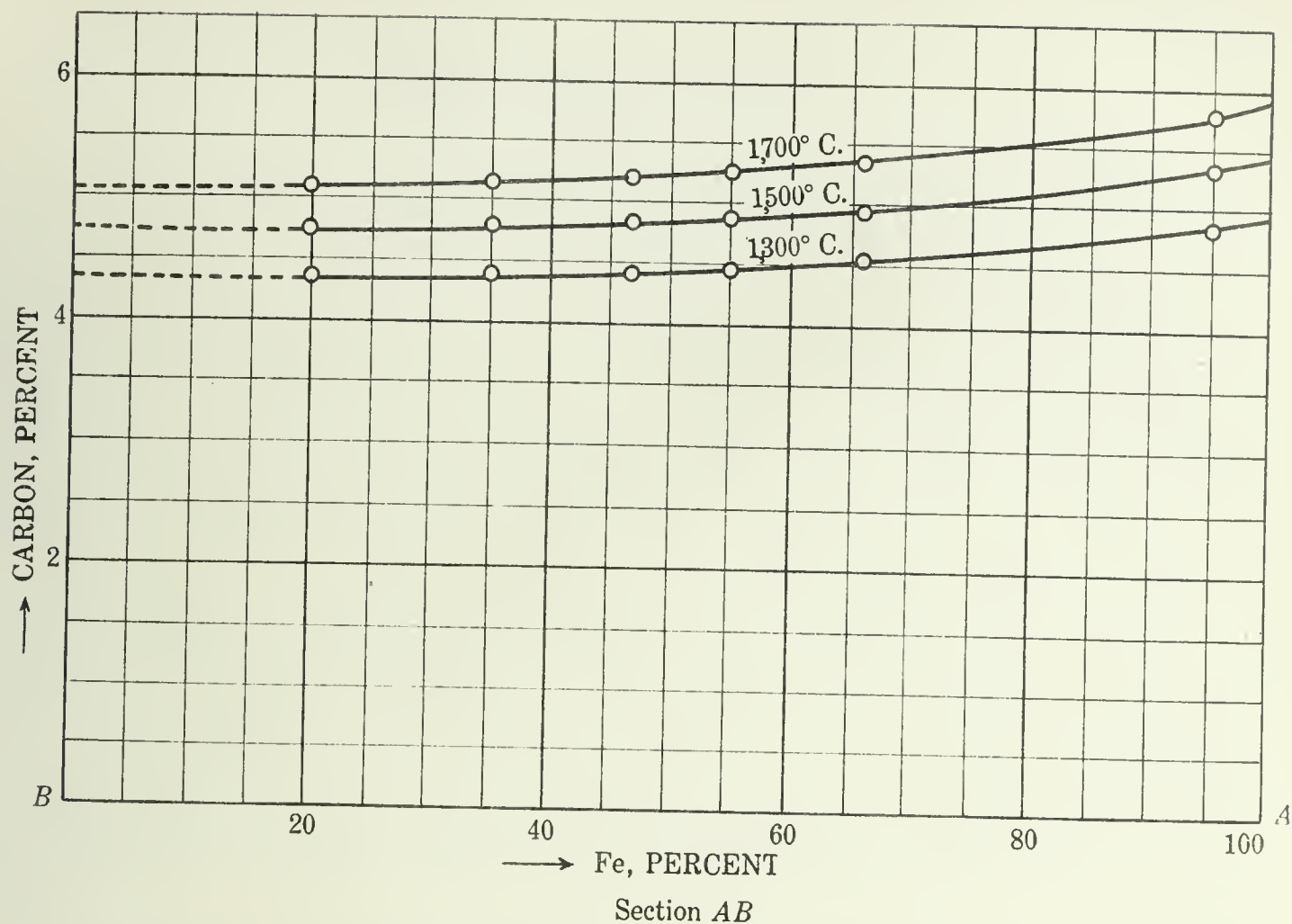


Figure 43.- Solubility in sections AB and AC of ternary system.

Manganese Reactions in Basic
Open-Hearth Furnace

From 1924 to 1929, when steel production increased to as much as 56 million gross tons per year, more than 80 percent was the product of the basic open-hearth furnace. During this period, also, the ratio of pig iron to scrap in the charge was reduced from approximately 1.0 to 0.6. In other words, the open hearth became a greater scrap-melting process, and it was found that higher contents of manganese in the pig iron were required. In consequence, the percentage of manganese in basic pig was increased from approximately 1.0 to 2.0 percent. Other factors causing this increase were the use of higher-sulphur ores and higher-sulphur coking and producer coal (390), and by using higher-manganese pig irons the open-hearth operator was insured lower sulphur contents of his finished product. This will be discussed later, on page 281.

When high-manganese pig iron is used higher residual manganese exists throughout the heat, reducing the amount of ferromanganese required at the end of the heat. This increase in the manganese content of pig iron was indicated by the increased concentration of MnO in basic open-hearth slags (151). Slags analyzed in 1917 were compared with those analyzed in 1927, and it was shown that the average manganese content of this type of slag increased from approximately 5 to 8 percent during this period. This loss in manganese through the slag medium has caused considerable comment concerning the conservation of manganese in this country, and this fact will be considered briefly later.

In the basic open-hearth process steel is made by melting scrap, large portions of liquid pig iron generally being added. In some localities where blast furnaces are not used in conjunction with the open-hearth department solid pig iron is charged with the scrap. The more general practice, however, is to add liquid pig iron during the first few hours of the heat. At this time a very complex slag is formed, which may be considered as essentially a calcium silicate containing variable proportions of MnO, FeO, MgO, and small amounts of Al_2O_3 , P_2O_5 , and sulphur, probably in the form of either or both calcium and manganese sulphides.

The chief reactions into which manganese enters during this process are as follows:



Because of the complex nature of the above slag exact determination of the characteristics of these reactions in the open-hearth process has baffled most attempts for solution. However, many phenomena involving manganese reactions may be qualitatively analyzed by studying the analyses of slag and metal samples taken from charge to tap of a specific open-hearth heat. Some of these phenomena will now be described.

In a certain heat in a 100-ton stationary open-hearth furnace complete tests were taken during the charging, refining, and tapping periods (195). The analyses of all samples are plotted in figure 44, and the variations with time are recorded therein. One hundred seventy-five thousand pounds of rail steel scrap were charged into the empty furnace, and as soon as possible liquid-steel samples were taken from the melted portion of the charge. Soon thereafter two liquid pig-iron charges were made, weighing 23,000 and 57,540 pounds, respectively, in the order added. The steel scrap contained 0.84 percent manganese, whereas the pig iron contained an average of 1.37 percent. Because the characteristics of manganese reactions can be explained only by plotting the variation of manganese content of the steel with time, the concentrations of carbon and silicon in the metal and of MnO , FeO , SiO_2 , and CaO are also shown in figure 44. After the second addition of liquid pig iron most of the scrap had melted, and the liquid portion of the charge contained approximately 1.1 percent manganese. It can be seen from the curve that the MnO content of the slag was greatly increased by the pig-iron addition, inasmuch as its value changed from 14 percent before the addition to 16.5 percent immediately thereafter.

From this time on, as represented by 12:20 o'clock on the figure, the carbon, manganese, and silicon contents of the liquid metal changed quite rapidly with time. Because of the great affinity of silicon for oxygen, the silicon content of the bath was decreased from 0.83 to practically 0.0 percent between 12:20 and 2:00. The carbon content also decreased very rapidly during this period, from 3.6 to 1.5 percent. The elimination of manganese was somewhat slower than either of these two materials, since it only decreased from 1.1 to 0.7 percent. The plotted MnO content of the slag is somewhat deceiving at this time. Although the manganese content of the metal decreased, the MnO content of the slag also decreased. This can be readily explained by the fact that at this time the slag volume was increasing rapidly. Not only did SiO_2 go into the slag due to the elimination of silicon from the metal, but also the limestone charged beneath the scrap at the beginning of the heat started to calcine and rise to the slag phase, thereby diluting the manganese oxide concentration. It is during this period that the so-called flush slags are obtained in open-hearth practice. The volume increases until the slag overflows the furnace bank at some determined spot. A discussion of the recovery of manganese from flush slags has been given on page 109. Elimination of 0.8 percent silicon in the charge would account for as much as 2,000 pounds of SiO_2 in the slag. Probably a similar amount of lime reached the slag phase during this period.

From 2:00 to 5:00 o'clock the manganese concentration in the metal decreased from 0.7 to 0.57 percent. Elimination of this amount of manganese in the metal was reflected in the increase in the MnO concentration of the slag from 10 to 15 percent. It was probably indicated in this manner, because the slag volume during this period was fairly constant. The carbon content of the metal decreased from 1.5 to 1.25 and held fairly constant until 5:00 o'clock. At this time two rather large additions of iron ore were made (a total of 7,000 pounds) to reduce the carbon content to specification limits. Immediately after these additions the manganese

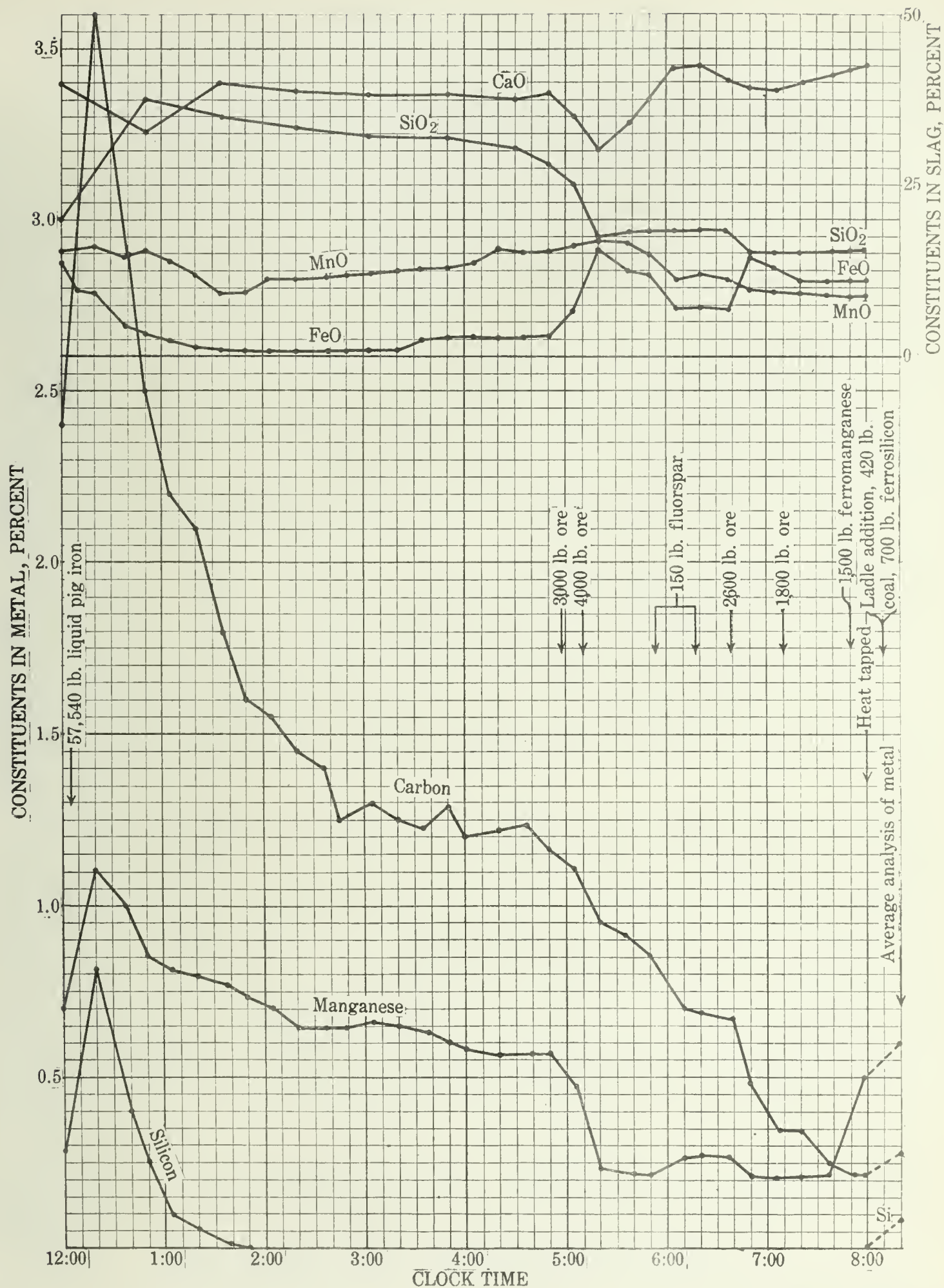


Figure 44.- Manganese reactions in the basic open-hearth furnace (Keats and Hertel)

and carbon concentrations dropped sharply from 0.57 to 0.24 and from 1.25 to 0.95, respectively. These ore additions were indicated in the analysis of the slag by reduction of the percentages of all constituents but FeO and MnO. The FeO concentration increased sharply; whereas the CaO and SiO₂ concentrations decreased rapidly. The concentration of the first oxide was obviously increased by means of the ore addition, whereas the reductions in CaO and SiO₂ were very probably due to dilution of the slag with iron oxide. In spite of this dilution the MnO concentration increased somewhat because of the oxidation of manganese in the metal.

During the next half-hour period between 5.45 and 6.15, 150 pounds of fluorspar were added to the slag. This material, as is well known, can reduce the viscosity of the slag to a considerable extent. The slag, previously very viscous, soon became quite liquid and either caused some lumps of lime to dissolve in the slag or eroded the walls of the furnace and thereby increased the CaO concentration of the slag. Again, the effect of dilution is indicated by a decrease in the MnO and FeO contents of the slag. For some unexplained reason the SiO₂ content held fairly constant.

At the end of this period an increase in the manganese content of the slag was found, which may be due to 1 or 2 factors or both, namely, a change in temperature, or the above-mentioned increase in the basicity (CaO content) of the slag. At 6.40 and 7.10, 2,600 pounds and 1,800 pounds of ore respectively were added. The effects of these additions on the analyses of the slag and metal are indicated only by an increase in the manganese content of the metal and the usual dilution effect of the slag constituents, namely, a decrease in all constituents other than FeO and an increase in the concentration of this material. A marked decrease in carbon content was also observed.

At this time the usual melter's break test was taken, and it was decided that the carbon content had reached the desired value; 1,500 pounds of ferromanganese were added at 7.52 and the heat tapped at 7.58. Tests taken during this short period indicated the expected rise in manganese concentration from 0.25 to 0.49 percent. No manganese in any form was added to the ladle. Instead, coal and ferrosilicon were added for recarburizing and deoxidizing. The average analysis of the heat, as determined by sampling various ingots, was: Manganese, 0.60 percent; carbon, 0.27 percent; and silicon, 0.08 percent. The increase in manganese observed between the tapping test in the furnace and the average analysis of the heat may be due to one of two things -- better distribution of the ferromanganese added just before tapping or the reduction of MnO in the ladle slag by means of the silicon and coal additions. It is very probable that both of these effects increased this value.

Various basic open-hearth heats have been described in a manner similar to the above, but more complete tests and more normal reactions of manganese were experienced in this heat than in various other published works.

Manganese Reactions in Acid Open-Hearth Furnace

In the acid open-hearth furnace the steel bath is in contact from charge to tap with a slag composed chiefly of MnO, FeO, and SiO₂. A rather complete

study (392) has been made of acid open-hearth slags, particularly those which have been cooled slowly. The mineral constituents of the slags after crystallization have been found to be tridymite (SiO_2), cristobalite (SiO_2), fayalite ($2\text{FeO} \cdot \text{SiO}_2$), and rhodonite ($\text{MnO} \cdot \text{SiO}_2$). The silica phases crystallize first on cooling, followed by fayalite and rhodonite, the latter two depending upon the ratio of FeO and MnO. When this ratio exceeds 2.7 fayalite is formed. In all other cases rhodonite is the constituent precipitated.

In virtually every heat on which analyses have been recorded the SiO_2 content of the slag varies little and is approximately 55 to 60 percent. The MnO and FeO contents seem to be more or less interchangeable and range from 10 to 20 or 25 percent of either constituent. The system FeO-MnO- SiO_2 (152) indicates that SiO_2 is saturated at $1,600^\circ \text{C}$. with either 35 percent MnO or 40 percent FeO. An equal mixture of FeO and MnO would then saturate SiO_2 at approximately 37.5 percent (FeO + MnO).

A rather complete heat has been described by De Marc (66), and the analyses are plotted in figure 45 against clock time. In this particular heat 141,000 pounds of material--35 percent pig iron and the remainder scrap--were charged to the furnace. The average analysis of the charge as calculated is given by the first points in figure 45. The manganese, silicon, and carbon contents of the charge were quite high in this particular instance, the average analysis being 1.75, 0.64, and 1.7 percent, respectively. During the melting period steel scrap was unquestionably oxidized rather thoroughly, and by the time all the charge had melted these constituents had been reduced appreciably, as shown in the figure, the manganese being down to 0.20 percent, the silicon to 0.07 percent, and the carbon to 1.15 percent. Immediately after melting down a small charge of limestone was added.

The iron-ore addition at 4:50 o'clock had little effect on the manganese and silicon contents but increased the rate of carbon elimination. The temperature at this time was quite high, because within a few hours the carbon continued to be eliminated in spite of the fact that both the manganese and silicon contents increased. This condition is quite normal in acid open-hearth practice and bears out the contention of various experimenters that at the higher temperatures the carbon reaction may continue, whereas the silicon and manganese reactions are reversed. The ore addition had little effect upon the iron oxide content of the slag, probably because the furnace lining was immediately eroded to such an extent that more slag was formed as ferrous silicate. This is borne out by the fact that the MnO content of the slag decreased, whereas the SiO_2 content increased at this time.

The reduction of manganous oxide in the slag increased until 9:10, when the oil and air were shut off and a slight increase in manganese oxide in the slag was indicated. This was probably due to the decrease in the silica content of the slag, as shown in the upper curve, resulting from a decrease in temperature and precipitation of solid silica from the iron-manganese silicate slag. At 9:45 a ferromanganese addition was made which increased the manganese content of the metal from 0.25 to 0.75 percent. The carbon content was increased slightly by means of this addition, due to the carbon content of

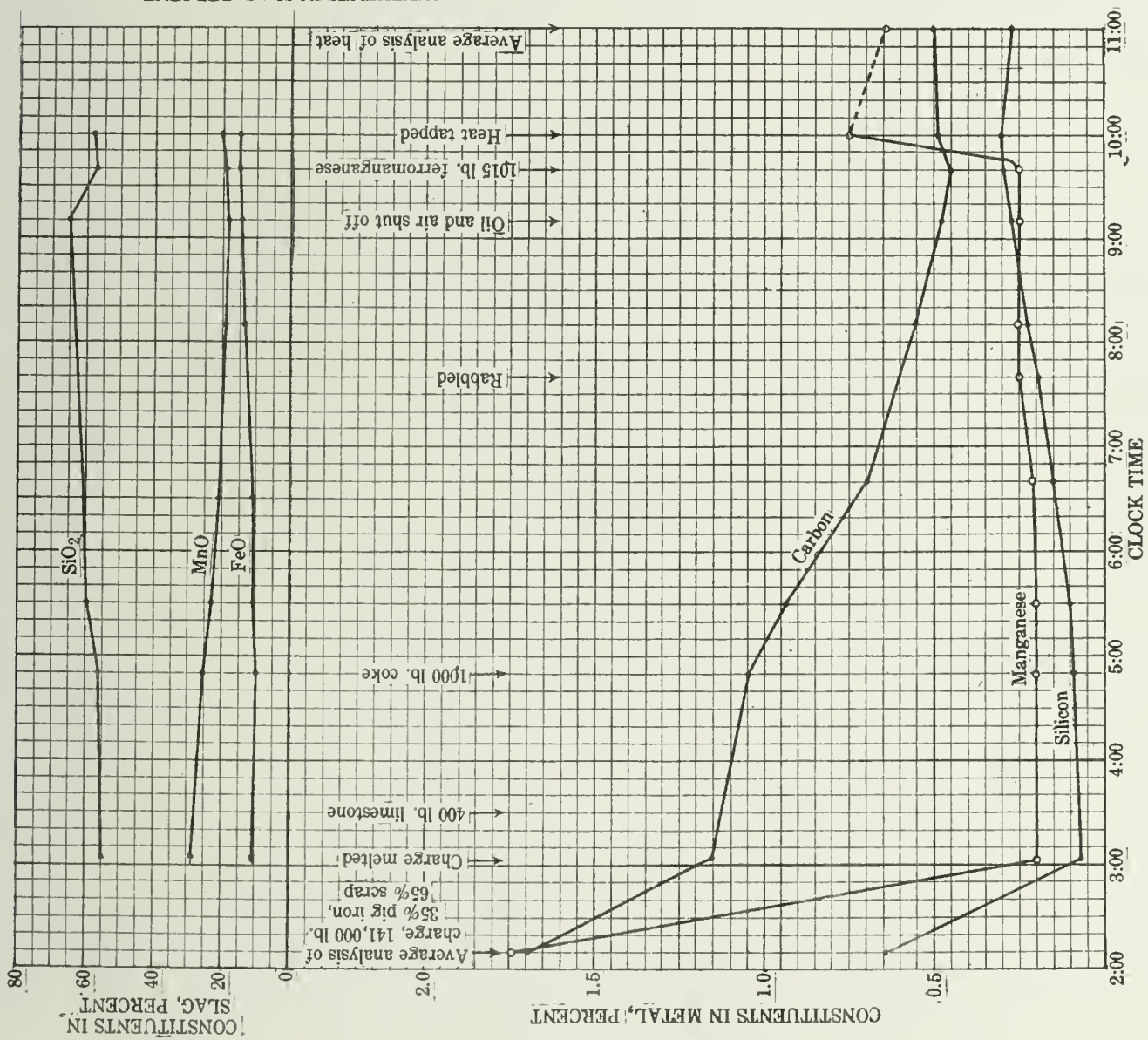


Figure 45.- Analysis of steel during acid open-hearth manufacturing process (De Mare).

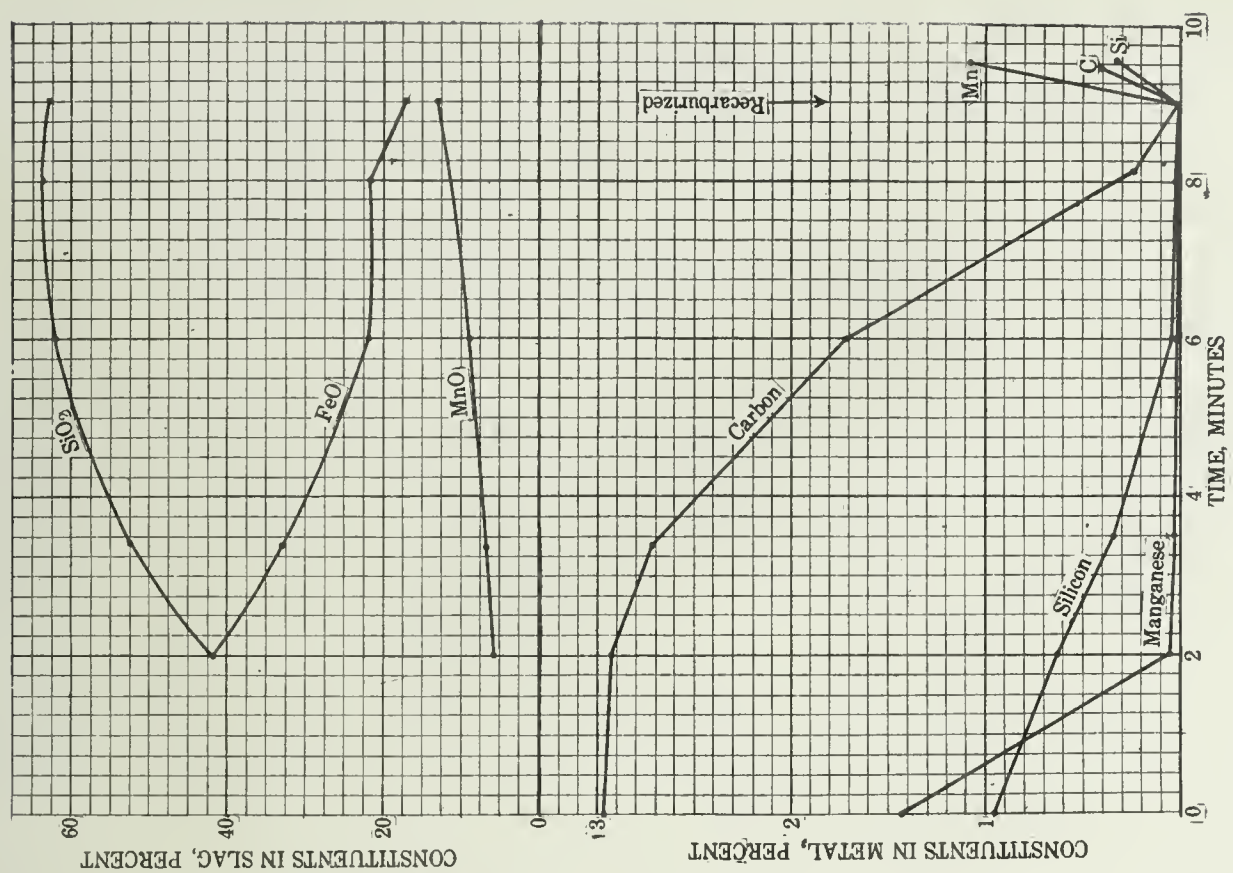


Figure 46.- Elimination of manganese, carbon, and silicon in acid bessemer process.

the ferromanganese. No silicon addition was necessary in this heat because by the time the ferromanganese addition was made 0.3 percent silicon had been reduced from the slag, which was more than enough for deoxidation. The average analysis of the heat was: Manganese, 0.64 percent; carbon, 0.49 percent; and silicon, 0.27 percent. These values represent a decrease in both manganese and silicon from the analysis of the tapping test, due to elimination of some MnO and SiO_2 in the form of manganous silicates from the metal to the slag.

In another acid open-hearth heat (263) manganese, silicon, and sulphur all decreased during the pouring period, which lasted 20 minutes. The manganese drop was larger than the equivalent drop of sulphur and it was suggested by these authors that some MnO probably had formed along with the MnS . More recent work (9) indicated that MnS is quite soluble in manganese silicate, and for this reason it is highly possible that the elimination of manganese, silicon, and sulphur as indicated above took place through the medium of manganese sulphide dissolved in manganese silicate inclusion particles. Such particles would be quite fusible and should readily flux with one another. The elimination in this connection should be relatively rapid.

The heat shown in figure 51 represents one of the best practices in the acid open-hearth furnace because of the high manganese content of the slag which was carried throughout the heat. In many American steel plants the FeO content is higher than the MnO . Better results have been indicated by the former practice, which has as one of its benefits the carrying of a higher residual manganese throughout the heat. Such a process requires less deoxidation.

Manganese in Acid Bessemer Furnace

The American practice in the acid bessemer furnace involves elimination of the constituents manganese, silicon, and carbon by blowing air through a bath of liquid pig iron. The American practice differs from that used in other countries in that low-silicon pig iron is used to diminish the "blow" or refining time. Just enough silicon (0.8 to 1.2 percent) is used to give sufficient heat from the oxidation of silicon to insure that no solidification of the metal is experienced during the carbon-elimination period. Fairly low-manganese pig iron is used in the American practice, the average manganese analysis being 0.40 percent. The English, German, and Swedish practices are much slower than the American, inasmuch as the silicon content of the pig iron is approximately 2.00 percent and the manganese also is somewhat higher -- 0.90 percent. Occasionally even much higher manganese and silicon contents than these are used in foreign practice. Regardless of the percentage of the constituents of the pig iron in all of the recorded acid bessemer heats, elimination of manganese preceded elimination of silicon. In every instance the manganese was virtually eliminated before the silicon was half removed. This is probably due to the acid lining, inasmuch as the formation of SiO_2 would be inhibited by the SiO_2 lining. Also, oxidation of the manganese probably is rapid in this case, because of the secondary reaction which involves the formation of a manganous silicate.

In Germany still another practice is popular. The temperature of a metal bath is allowed to increase until the carbon is eliminated in preference to the manganese and silicon. In this manner a low-carbon steel containing appreciable amounts of residual silicon and manganese is made. The reason for this practice is that it is much easier to obtain a desired specification, and further recarburization and deoxidation in the ladle are not required. In one instance to be found in the literature (361,p.106), at the end of the blow the carbon had been eliminated to 0.1 percent, the silicon to 0.34 percent, and the manganese to 0.60 percent by this type of procedure. No additions were made to the ladle because these values were within the desired specifications for the steel.

An example of the relative elimination of manganese with regard to the elimination of silicon and carbon is shown in figure 46. In this particular heat (361,p.106) a liquid pig iron was charged which contained: Carbon, 2.96 percent; manganese, 1.42 percent; and silicon, 0.94 percent.

Within 2 minutes after the blow was started, virtually all of the manganese had been eliminated, being reduced to approximately 0.1 percent. Virtually no carbon had been eliminated and only about 25 percent of the silicon, the latter being reduced to 0.62 percent. By this time a slag had formed that contained MnO and FeO from the oxidation of the iron and manganese by the air bubbling through the liquid steel. Also, these two oxides had eroded the converter lining to such an extent, and enough SiO₂ had been formed by oxidation, so that approximately 40 percent was present in the slag. During these first stages the FeO content of the slag was quite high, as shown, but soon was diluted by the above fluxing procedure until only 20 percent of FeO was present in the slag. By this time virtually all the silicon in the metal had been eliminated. The passage of more air through the vessel was required, however, to remove the remainder of the carbon. This was done within a 3-minute period between 6 and 9 minutes plotted on the figure. The MnO content of the slag remained fairly constant throughout the heat (6 to 8 percent) until tapping, when spiegleisen was added to the ladle. At this point the MnO content of the slag increased to 13 percent. The analysis of the steel was affected as the diagram indicates. The spiegel addition to the ladle raised the contents not only of manganese but also of carbon and silicon. Virtually all of these constituents were very low at the time of the addition, and the final analysis was: Manganese, 1.15 percent; carbon, 0.45 percent; and silicon, 0.38 percent. As has been explained in the previous sections, spiegel is an alloy of iron, manganese, silicon, and carbon, each in sufficient proportions to affect the analysis as indicated.

Manganese in Basic Bessemer Process

Although no basic bessemer operations are being conducted in the United States at present, the process is interesting from the standpoint of manganese metallurgy when contrasted with the acid process so popular in this country.

Wedding (379) described a heat by Finkencor at the Rhine Steel Works in 1833 which seems to typify other basic heats made and described since that time.

As would be expected, the manganese is not eliminated as readily in this practice as in the acid furnace. (See fig. 47.) The silicon oxidizes first to SiO_2 and is then carried to the slag to become a constituent part of the calcium silicate. During this stage the manganese is eliminated slowly, and after 8 minutes of the blow holds fairly constant at 0.14 to 0.18 percent, while the carbon blow or elimination is taking place. When most of the carbon has been eliminated the manganese again oxidizes, as shown by the small value (0.01 percent) indicated after 15 minutes and 13 seconds of the blow. Subsequently only a trace of manganese remains.

The manganous oxide in the slag decreased (as indicated in fig. 47) after 5 minutes and 21 seconds of the blow, although some of the manganese in the metal was still being eliminated to the slag. This effect was probably due to the fact that the lime usually charged in the bottom of the furnace came to the slag surface after 5 minutes and took some time to dissolve all the lumps in the slag. During this process the manganous oxide was diluted, and its concentration decreased.

The chief reason for using the basic practice is to remove the phosphorus from high-phosphorus pig irons. This is accomplished by means of the afterblow. The liquid iron is allowed to cool some time after the silicon, manganese, and carbon have been eliminated. The secondary blow is then started, which involves the oxidation of the phosphorus to P_2O_5 , followed by the formation of $3\text{CaO} \cdot \text{P}_2\text{O}_5$ in the slag. Some writers also contend that $(\text{MnO})_x (\text{P}_2\text{O}_5)_y$ forms during this period. In this particular heat the afterblow was started at 15 minutes 13 seconds after charging and lasted until 19 minutes 49 seconds, during which the phosphorus content of the metal was decreased from 2.000 to 0.087 percent.

Some conception of the manner in which manganese reacts during the refining of steel in the basic bessemer process was obtained by Faust (90). This investigator obtained slag and metal samples from various heats in which the manganese content of the liquid pig iron charged was increased. A relation was established between the ratio of manganese to iron in the slag and the manganese content of the pig iron, and it was found that the ratio increased from 0.4 to 0.7 by increasing the manganese content of the pig from 0.6 to 1.8 percent.

It was also found that the amount of manganese oxidized in the basic bessemer remained fairly constant at 85 percent, although the manganese content of the refined metal increased from 0.15 to 0.25 percent by means of the same increase in the manganese content of the pig iron.

Another relation was established between the manganese content of the refined metal and the ratio of MnO to FeO in the furnace slag and is expressed as follows:

$$\text{Percent Mn in metal} = 0.4 \times \frac{\text{Percent MnO in slag}}{\text{Percent FeO in slag}}$$

As shown above, the ratio of these constituents in the slag varies from 0.4

to 0.7, and in accordance with this the manganese content of the metal would necessarily range from 0.16 to 0.28.

Manganese Reactions in Basic Electric-Arc Furnace

The most common melting practice in the basic electric furnace involves charging cold scrap to the furnace, melting down under an oxidizing slag, and removal of this first slag followed by preparation of a synthetic slag. The synthetic slag is prepared from lime, fluorspar, and coke and is usually identified by the term "carbide slag". The purpose of this secondary slag is to prevent oxidation by recarburization and stabilizing of the analysis of the metal so that time may be allowed for an analysis to be received from the laboratory. Variations in analyses for carbon, manganese, and silicon in a typical ball-bearing steel heat (349) are shown in figure 48. The manganese reactions in this heat are quite simple, as can be readily seen from the curve. Manganese in the scrap charged in the furnace was approximately 0.40 percent and during melting under the oxidizing slag was reduced to approximately 0.25 percent. Shortly after the melting down the first slag was raked off and a carbide slag put on, and a slight increase in the manganese content of the metal is noted. If the first slag had been entirely removed in this heat, it is quite doubtful if any increase at this time would have been indicated, inasmuch as there would not have been any source of manganese other than that already in the metal bath. At 9:15 clock time on the chart, ferromanganese was added to the heat, thereby increasing the manganese content to 0.35, at which concentration it remained for the remainder of the heat. Some increase in the carbon content was noted, which was probably due to both the carbon in the ferromanganese and that in the ferrochrome, which was also added at 9:15. At 9:50 wash metal was added to the bath to increase the carbon content of the metal without increasing the manganese. Wash metal, as it is known, is essentially an iron-carbon alloy and contains very small amounts of manganese and silicon. The effect of the ferrosilicon addition on the silicon content of the metal is indicated clearly on the chart. The slag analyses plotted in the upper portion of figure 48 indicated that there was little or no manganese in the carbide slag. The first or oxidizing slag contained approximately 5 percent MnO , but this was removed by slagging at 8.34. The manganese content of many low-alloy and plain-carbon tool steels is quite low compared with steels made by other processes. The ladle analysis of the heat just described indicated the presence of 0.34 percent manganese. Various other specifications for steels of this type require a final manganese analysis of 0.25 to 0.45 percent. As can be seen from the description of the above heat, the only attention given manganese in the electric furnace is a small addition of ferromanganese toward the latter part of the heat and under the carburizing slag. The loss of manganese under these conditions is slight.

The basic electric furnace is also used for preparation of high-manganese steels, such as the austenitic steels containing as much as 10 to 15 percent of this constituent. In the preparation of these materials in the basic electric furnace a somewhat higher residual manganese is carried throughout the heat, followed by large additions of sometimes both low- and

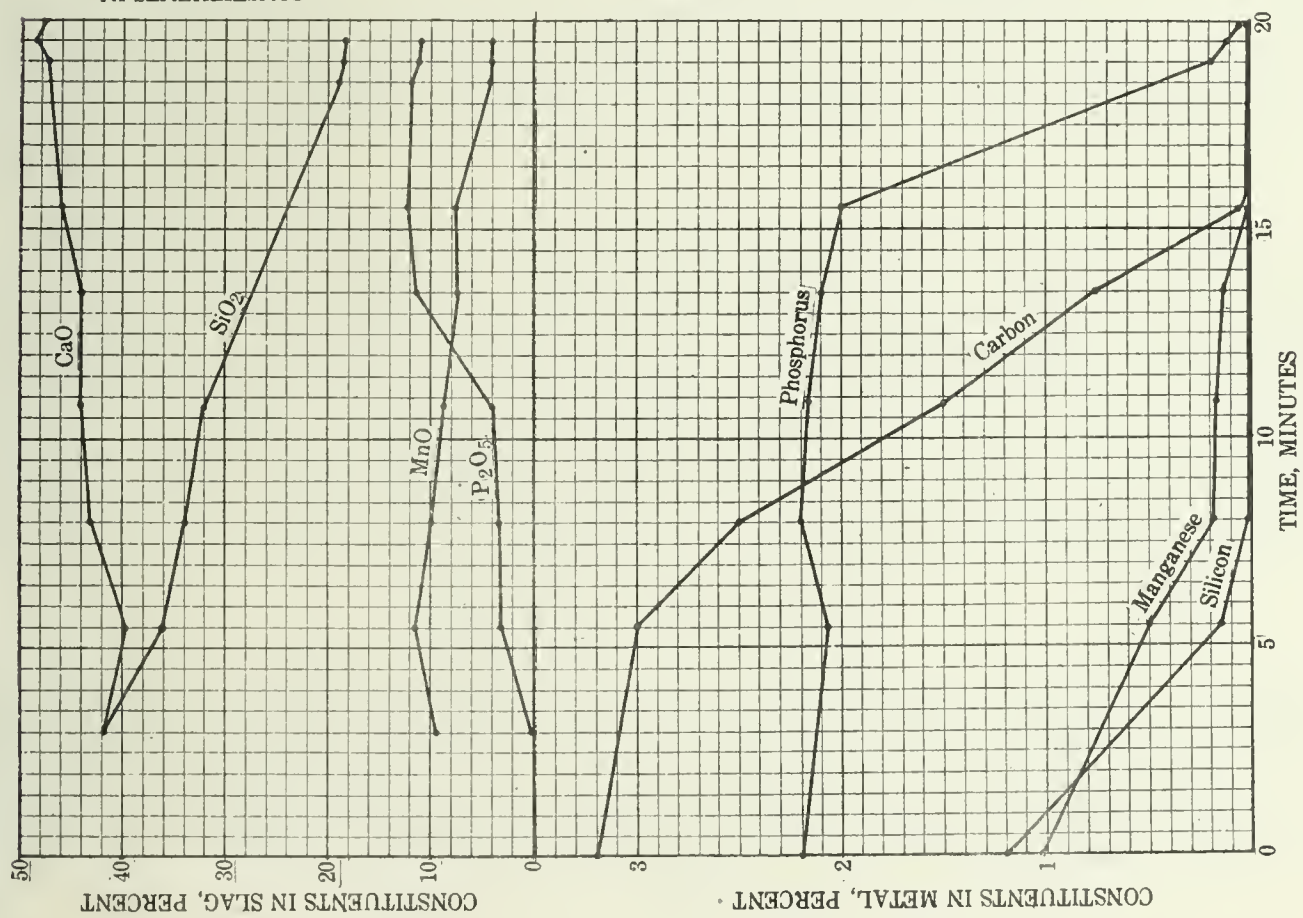


Figure 47.- Manganese reactions in basic bessemer process (Finkener).

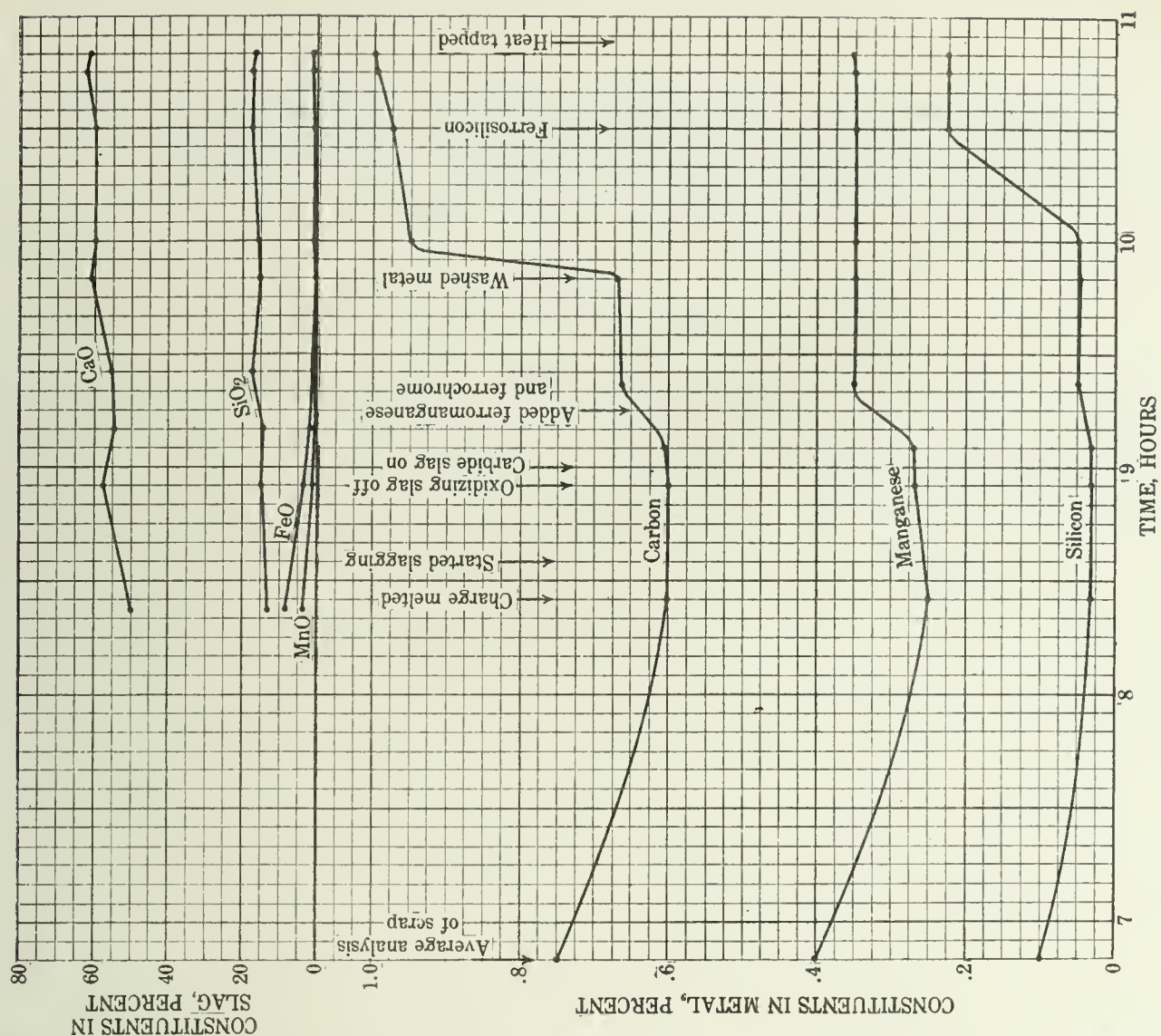


Figure 48.- Manganese reactions in basic electric arc furnace (Sisco).

high-carbon ferromanganese. In this instance also only slight losses of manganese are experienced because of the carbide slag. If accurate analyses of melt-down tests are given to the melter he has no difficulty in meeting the desired specifications. A more complete description of these steels will be given later.

Manganese in Coreless Induction Furnace

Little work has been done toward refining steel in the high-frequency induction furnace, principally because of the vigorous convection currents set up in the liquid steel. Recent developments, however, have shown several ways in which these currents may be reduced considerably, and future investigations are expected to involve studies of various types of basic and acid slags for the purpose of refining steel in this type of furnace.

Under present conditions the high-frequency furnace is used almost exclusively for remelting processes. Oxidation reactions occur but are due largely to the atmosphere above the moving metal bath, and in many instances these are reduced by large additions of some deoxidizer, such as ferrosilicon, immediately after melting down. If no deoxidizer is added some of the oxide-forming elements already in the steel scrap are oxidized and eliminated to a slag which forms a rim at the top of the crucible. Because this rim of slag is out of the high-temperature zone it is usually solid and does not appreciably affect the reactions. The oxidation of steel by the atmosphere in this type of furnace has been studied (389) and the results may be briefly described as that in a basic-lined furnace (magnesite) oxidation of silicon from the scrap is much more rapid than oxidation of manganese. The opposite is true in the acid-lined furnace, inasmuch as the oxidation product SiO_2 is already present in the refractory lining. This of course holds back the silicon reaction. Similar differences in velocities of reactions have been noted in previous sections in which acid and basic open-hearth heats were compared.

One author has added as much as 2 percent manganese to such steel, but the CO blowholes persisted in the solidified material. This is also the experience of Korber (211). In fact, it has been pointed out (94) that the C-FeO reaction is slightly endothermic at steel-making temperatures ($1,600^\circ\text{C}$), whereas the Mn-FeO reaction is slightly exothermic. Some indications show that at $1,583^\circ\text{C}$ carbon and manganese may have equal tendencies to form their oxides, whereas above this temperature carbon reacts more strongly than manganese. Below this temperature the reverse is true, in accordance with the recommendation that high temperatures result in manganese savings.

Manganese in Wrought Iron

In the old puddling process for making wrought iron cold pig iron was usually charged into a reverberatory-type furnace. The charge was usually melted within half an hour, and as soon as possible iron oxide was added to remove the various metalloids of the liquid pig iron. Within 10 to 20 minutes the manganese content had been reduced 90 percent. With elimination

of carbon, silicon, and manganese the melting point of the iron was raised, and solidification or "graining" began. The puddler soon started to separate the granulated material into several lumps, which he kneaded into small balls to squeeze out some of the slag formed through reaction of the above constituents with iron oxide. All these lumps of granulated metal were then worked together into one large lump and finally withdrawn from the furnace and sent through a rotary squeezer that removed a considerable portion of the remaining slag. The manganese remaining in the wrought iron thus formed (0.05 percent) is probably present as MnS , MnO , and some small amounts of the alloy manganese.

This procedure has recently been supplanted by the Aston (11) process, which essentially involves refining of liquid pig iron in the acid bessemer and pouring of this material into a ladle of synthetically prepared ferrous silicate slag. The manganese reactions involved are the same as those previously mentioned in the description of the acid bessemer process. It can be seen that virtually all the manganese in the original pig iron is eliminated.

Manganese Reactions in Liquid Steel

Because of the important economic situation with regard to manganese considerable research has been done with the hope of determining the various equilibria of manganese reactions in liquid steel. Many such attempts have been made without consideration of the complicated furnace slags.

Manganese -- The Deoxidizer

Manganese has been overrated as a deoxidizer for steel. Many authors discuss the deoxidation of steel with manganese as though the reaction is quite strong, and no further deoxidation is necessary if manganese is added. The strength of a deoxidizer is reflected in its ability to inhibit the reaction between carbon and iron oxide. In other words, it should be a better deoxidizer than carbon. In this regard manganese is only slightly better than carbon.

In making low-carbon rimming steel, 0.20 to 0.30 percent manganese is added to the furnace and ladle in the form of ferromanganese or spiegel. No other deoxidizer is employed. When this material is poured into the molds enough carbon monoxide escapes to give the appearance of boiling. The solidified ingot has the well-known structure of rimmed steel, characterized by rows of blowholes equidistant from the ingot surface.

Giollitti (113) states that steel may be deoxidized to the point of inhibition of the $C-FeO$ reaction by allowing enough time for the nearly colloidal product of the reaction MnO to rise from the steel bath into the slag. The manganese content must be maintained constant during this period, however, and sometimes several hours are required to produce thorough deoxidation. This idea has been consolidated in a recommended steel practice for the basic open hearth, which, as quoted from Giollitti, is as follows:

As soon as melting down is complete the bath should contain at least 0.60 percent carbon and 0.25 percent manganese. Only when these percentages are exceeded can small additions of ore be made with the view to shorten the first phase of the operation. Additions of ore must absolutely cease after the carbon has fallen below 0.55 percent and the manganese below 0.20 percent. When the carbon has reached 0.40 percent, successive additions of ferromanganese are made, adding each time about 0.25 percent manganese (referred to the total charge) so as to always maintain in the bath a percentage of manganese between 0.20 and 0.45 percent. In this manner the carbon is allowed to fall very gradually until it is brought down to 0.20 percent.

In one heat recorded, five additions of ferromanganese were made. This represents a costly but standard practice of a large European company in making high-grade steel.

Attempts have been made to determine the equilibrium in the reaction $\text{FeO} + \text{Mn} \rightleftharpoons \text{MnO} + \text{Fe}$, in which manganese is dissolved in the liquid iron phase and the slag is composed of only FeO and MnO. These attempts have not been very successful, particularly because of refractory difficulties. A pure slag containing only these two oxides is very erosive on practically any type of acid or basic refractory material.

Other investigators have attempted to determine the effects of high concentrations of CaO and SiO_2 in the FeO-MnO slag phase. Many experimental difficulties have been experienced by this latter group of workers, as would be expected, inasmuch as analyses of open-hearth slags vary so widely. As was pointed out previously, an open-hearth slag is essentially a calcium silicate in which iron and manganese oxides are supposedly dissolved. In every investigation the assumption has been made that the reaction given above is homogeneous and that both FeO and MnO are at least slightly soluble in the slag and metal phases.

Calculations of Equilibrium for Reaction $\text{FeO} + \text{Mn} \rightleftharpoons \text{MnO} + \text{Fe}$. - Probably the first investigation of this equilibrium was made by Le Chatelier (231). His work involved the calculation of the dissociation pressures of FeO and MnO from the Carnot-Clapeyron equation. The pressure values calculated by this author were:

$$\begin{aligned} p_{\text{O}_2}(\text{FeO}) &= 10^{-8} \text{ atmospheres for reaction } 2\text{FeO} \rightleftharpoons 2\text{Fe} + \text{O}_2, \\ p_{\text{O}_2}(\text{MnO}) &= 10^{-14} \text{ atmospheres for reaction } 2\text{MnO} \rightleftharpoons 2\text{Mn} + \text{O}_2. \end{aligned}$$

Le Chatelier made these calculations by assuming that manganese and iron are completely miscible at steel-making temperatures and that the saturation concentration of FeO in liquid iron is 1.1 percent, and also that MnO is nearly insoluble in liquid iron. By combining the above two values,

$$K_{\text{II}} = \frac{p_{\text{O}_2}(\text{FeO})}{(\text{per cent FeO}_{(\text{sat.})})^2} = \frac{10^{-8}}{1.21}$$

and $K_{III} = [p_{O_2} (MnO)] \cdot [\text{percent Mn}_{(sat. = 100)}]^2 = 10^{-14} \times 10^4$,

and $\frac{K_{III}}{K_{II}} = 0.0121$ for a bimolal reaction,

and $K_I =$ equilibrium constant for the product $Mn \times FeO = 0.0121 = 0.11$.

Styri (364) and McCance (249) calculated this constant in an identical manner, but used different values for the percent saturation of ferrous oxide in iron and for the dissociation pressures of the two oxides as calculated from the Nernst heat theorem. Their values are respectively 0.126 and 0.111. These investigators assumed throughout that manganous oxide was nearly insoluble in iron at $1,600^\circ C$.

Oberhoffer and Schenck (279) were not satisfied with these calculations and proceeded to make some experimental determinations. These workers pointed out the probability of manganous oxide having an appreciable solubility in iron and wrote their mass-action equation as follows:

$$K_1 \left(\begin{array}{c} \text{all materials} \\ \text{dissolved in iron} \end{array} \right) = \frac{FeO \times Mn}{MnO}$$

instead of the usual $(Mn \times FeO = k)$ expression.

Also, they considered the function of the liquid-slag phase above the liquid metal as representing a system of two immiscible liquids in and between which the MnO and the FeO were dissolved and distributed according to the Nernst distribution law.

In other words

$$L_1 = L_{FeO} = \frac{FeO(slag)}{FeO(metal)} \quad \text{and} \quad L_{MnO} = \frac{MnO(slag)}{MnO(metal)} = L_2$$

where L_{FeO} and L_{MnO} are the distribution coefficients.

Then with these ratios in mind they constructed a constant as follows:

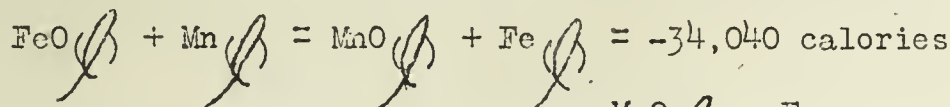
$$\frac{1}{K_2} = K_1 \times \frac{L(FeO)}{L(MnO)} = \frac{FeO(slag) \cdot Mn(metal)}{MnO(slag)}$$

They then determined the value of K_2 to be $2.66 = \frac{(MnO)_{slag}}{(FeO)_{slag} \times (Mn)_{metal}}$.

No attempt was made by these investigators to determine the variation of K_2 with the temperature, and no values were obtained for the relation between MnO and FeO in the metal and their variation with the manganese content. Also the percentage of manganese (metal) as given by them is composed of both free manganese and manganese as MnO (or MnS), and the constant is probably smaller because of this correction.

Benedicks and Lofquist (22) combined the work of previous investigators and calculated the variation of total oxygen with the manganese content of the iron by assuming that all combinations exist in the system iron-oxygen-manganese.

An attempt (95) has been made to calculate the equilibrium constant for the Mn-FeO reaction from available data, such as specific heats, heats of formation, fusion, etc. More reliable data are available now, and a recalculation should be made. The free energy was calculated for the reaction



or

$$K = 9550 = \frac{\text{MnO} \times \text{Fe}}{\text{FeO} \times \text{Mn}}$$

If FeO and MnO are both soluble in liquid iron, as assumed by Fitterer, the equilibrium constant should be modified accordingly.

The following constant was constructed:

$$K = \frac{\frac{\text{Percent MnO}_{(e)}}{\text{Percent MnO}_{(s)}} \times \text{percent Fe}}{\frac{\text{Percent FeO}_{(e)}}{\text{Percent FeO}_{(s)}} \times \text{percent Mn}_{(f)}}$$

in which the subscripts e and s denote "equilibrium" and "saturated" respectively; Mn_(f) is the uncombined manganese in the iron determined by subtracting the manganese present as MnO from the total manganese. The MnO content of the iron was determined by the electrolytic method. (95).

At 1,600° C. this constant was found to be 512. The saturation of MnO in the iron was found to be 0.333 percent, whereas the saturation of FeO at the same temperature (155) is 1.25 percent.

Some of the experimental work used in the determination of this constant was admittedly in error; but the electrolytic method offers a new solution to this problem, and it is hoped that future work will continue in this direction. It is particularly important to accurately determine the solubilities of MnO in iron at various temperatures. Unless this is done further consideration of this reaction must be postponed.

The most recent work on this reaction by Körber (211) involved the study of constant $K = \frac{\text{MnO} \times \text{Fe}}{\text{FeO} \times \text{Mn}}$, in which the oxides are in the slag phase. The

constant was shown to range with temperature from 60 at 1,950° C. to 260 at 1,530° C. or the melting point of iron. Corrections were made for the solubility of FeO in iron but it was assumed that MnO was nearly insoluble, and

no correction was made in this regard. It is to be regretted that no determinations were made of the MnO content of the metal. The samples prepared were of such a nature that this very important factor could have been determined once for all.

In this work, as in the others discussed above, no mention was made of the effect of complex open-hearth slags upon the equilibrium constant. The only practical instance in which these constants would apply occurs during the deoxidation of steel with a manganese alloy. If the saturation of MnO is exceeded, MnO particles thereof will be distributed throughout the bath. FeO is entirely miscible in MnO and FeO-MnO particles of these oxides would soon be formed. The metal surrounding these particles would then tend to approach equilibrium with the FeO-MnO slag.

In nearly all other practical cases the metal in which manganese, FeO, and possibly MnO are dissolved approaches equilibrium with a slag in which FeO and MnO are only minor constituents. The effects of the various slags on the equilibrium have long been a matter of speculation.

Characteristics of Manganese Deoxidation Determined from Open-Hearth Data -- Effect of Basic Slags. - Feild (92) calculated the constant $m = \frac{\text{MnO}}{\text{FeO} \times \text{Mn}}$ to be about 0.01 from metal analyses of a certain open-hearth heat. His MnO values were determined by calculating the decrease in FeO caused by known additions of manganese. This procedure does not indicate that equilibrium was established and must therefore be discounted. Likewise, no indication of the change of m with temperature was given.

Colclough (55) lists 10 values for this constant without temperature consideration and obtains an average value of about 1×10^3 . Colclough, however, made one statement which expresses an observation made above -- namely, that no addition of manganese alone will completely inhibit the evolution of CO gas formed from the reaction of carbon with FeO.

Herty (150) combined the various relations previously determined by Keats and himself (195) in such a manner that a very useful equilibrium constant and its change with temperature were obtained. Certain corrections were necessary, and a later publication (156) by the same author discusses the constant as it stands at present.

The relationship selected by this investigator was the reciprocal of Oberhoffer's expression (equation A, p. 276) and is expressed as follows:

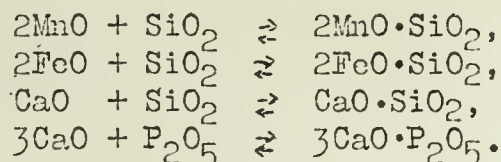
$$K' = \frac{(\text{MnO})_{\text{slag}}}{(\text{FeO})_{\text{slag}} \times (\text{Mn})_{\text{metal}}}$$

wherein $(\text{MnO})_{\text{slag}}$ and $(\text{FeO})_{\text{slag}}$ are expressed in mole fractions and $(\text{Mn})_{\text{metal}}$ is expressed as weight percent.

The amounts of MnO and FeO in the metal phase were assumed to be proportional to those of the slag phase, according to the two distribution coefficients (L_1 and L_2) which were introduced by Oberhoffer.

Also, it was assumed that the MnO and FeO in the slag were probably associated in some manner with the acid constituents SiO_2 and P_2O_5 , which are also present.

The extent of this association was likewise assumed to depend primarily upon the over-all balance of the following reactions:



From the heats of formation it was assumed that the calcium silicate was probably the most stable compound and that the amounts of free MnO and free FeO were indirectly proportional to the amount of the free or the available base CaO. For these reasons, a correction factor (mole fraction AB) 0.5 was applied to K' and the present constant being used for open-hearth calculations is:

$$K'' = \frac{(\text{N}_{\text{MnO}})_{\text{slag}} (\text{N}_{\text{A.B.}}) 0.5_{\text{slag}}}{(\text{N}_{\text{FeO}})_{\text{slag}} (\text{percent Mn})_{\text{metal}}}$$

where N = mole fraction.

The variation of this corrected constant with temperature was determined by means of Van't Hoff's isochore by assuming that the heat of reaction is constant in the temperature range considered. The expression developed was:

$$\log_{10} K_{\text{Mn}} = \left(\frac{10^5}{T} - 52.1 \right) \frac{1}{8.88}.$$

Later the opinion was expressed that if the mole fraction of available base in the open-hearth slag is less than 0.3 the following expression can be used:

$$\log_{10} K_{\text{Mn}} = \left(\frac{10^5}{T} - 50.27 \right) \frac{1}{6.27}.$$

In general, this expression seems to apply to open-hearth practice, particularly at such times when the rate of elimination of manganese is low, or, in other words, when the reactants are approaching stable conditions.

Schenck (337) made many calculations similar to Herty's on the available base concept. The various compounds such as the calcium silicates and their influence were assumed from various angles but in general brought this very complex problem no nearer satisfactory solution.

Manganese Oxide in Steel. - As stated in the discussion of manganese reactions, some metallurgists believe that MnO resulting from the Mn-FeO reaction is entirely insoluble in steel. Others hold that inasmuch as the deoxidation reaction is reversible this oxide must be at least slightly soluble in liquid iron, otherwise the reaction would go to completion, and no manganese would remain in the steel.

Fitterer (94) melted electrolytic iron under MnO and held the melt at $1,600^{\circ}$ C. The slag of course changed to an FeO-MnO mixture, but the metal was later found to contain 0.06 percent MnO by annealing and electrolytic extraction. This test indicates that MnO may be partly soluble in liquid iron, which is in equilibrium with an FeO-MnO slag. If enough manganese is added to steel so that the solubility of MnO is exceeded, MnO particles will be precipitated. These particles will also absorb some FeO from the surrounding steel.

Daniloff (62) found that FeO and MnO are miscible in all proportions, and the liquidus and solidus of the system FeO-MnO are regular curves leading from $1,360^{\circ}$ C. at the melting point of pure FeO to $1,600^{\circ}$ C., the melting point of pure MnO. This work has since been checked by Andrew (9). Precipitated MnO particles would then assume compositions designated by this diagram for temperatures within the above range.

Pure MnO crystals have been found (94) in iron to which 2 percent manganese had been added. These are reproduced in figure 49. Dendrites of MnO have also been found in the same samples as shown in this figure. Similar materials were found in steels deoxidized with 1.76 percent manganese by Körber (211).

MnO inclusions resulting from lesser additions of manganese sometimes are homogeneous solid solutions of FeO and MnO. Occasionally, however, they have the appearance illustrated in figure 50. It is believed that this type of particle rapidly passed through the region of solid-liquid immiscibility in the FeO-MnO system, and time was not allowed for homogenization. Otherwise, a homogeneous particle would have been observed. This structure appears in the large FeO-MnO inclusions and seldom in the smaller particles.

The particles of MnO or MnO-FeO described above assist greatly in removing more insoluble and refractory materials, such as SiO_2 , from liquid steel. For example, low-melting particles of manganous silicates are formed when manganese-silicon alloys (152) varying from 4:1 to 8:1 in ratio of manganese to silicon are added to an oxidized liquid steel. These particles are fusible enough to coalesce whenever they come in contact.

It has been suggested that the mechanism of this action involves the formation of insoluble SiO_2 particles and soluble MnO (and possibly some MnO particles). MnO is absorbed into the SiO_2 particle so as to approach a slag-metal equilibrium, the slag in this case being numerous heterogeneously dispersed particles. As the manganese silicate particles are formed they rise toward the furnace slag surface, coalescing with each other until the majority have been eliminated from the metal bath. This requires a very

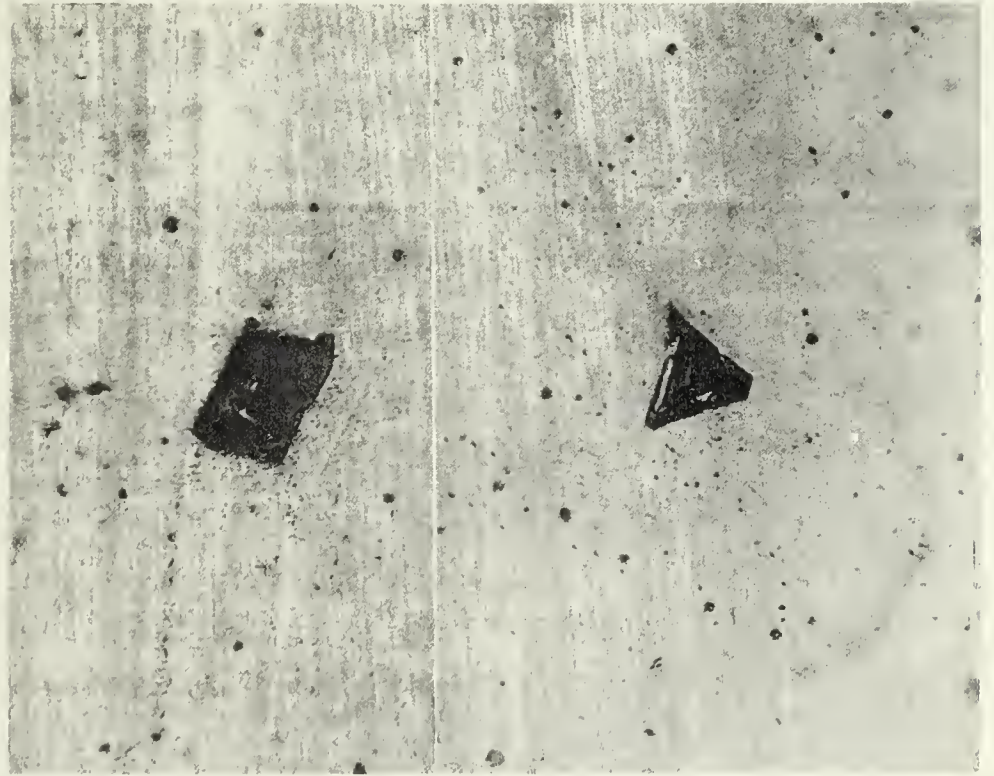


Figure 49.-- MnO crystals in steel deoxidized with manganese.



Figure 50.-- FeO-MnO inclusion in steel partly deoxidized with manganese.

short time, and the steel is left in the furnace only 5 to 10 minutes after the alloy addition.

This fluxing action of MnO is probably the basic reason for preparation of high-quality steel as suggested by Giollitti (113). The manganese-silicon alloys are probably more economical than this practice, however, and much less time is required for the "cleansing" action. This process has been discussed in detail on page 288.

Manganese -- Desulphurizer

As stated in the introduction to this section, the desulphurizing action of manganese in steel was predicted by the chemist Henderson (106) during the application of ferromanganese to English steels.

Soon after, Hackney (133) suggested that four times as much manganese should be added to steel as there is sulphur present; otherwise, difficulty would be encountered in rolling or forging the product. Stead (354) determined that the ratio of manganese to sulphur should be at least 8:1 to obtain the best beneficial effects of manganese.

MnS is less soluble in liquid iron and more soluble in slag than is FeS, and if given time some of these particles will rise and enter the slag phase, resulting in elimination of sulphur. According to McCance (250) 1 percent of manganese in pig iron will reduce the sulphur content 45 percent during the period between the tapping of the blast furnace and pouring from the mixer. An increase to 1.2 percent manganese will reduce the sulphur content 50 percent.

In later tests by Wheaton (402, pp. 1112-1114) the manganese was raised to as much as 2 percent, and the sulphur content of the pig iron was reduced almost 60 percent. Liquid pig iron has been held for considerable periods of time (153), and calculations were made of the residual manganese and sulphur remaining in the iron after virtually all of the manganese sulphide which had formed had been eliminated from the metal bath. A desulphurizing constant was calculated from these tests and was found to be:

$$K = (\text{percent manganese}) (\text{percent residual sulphur}) = 0.070$$

at approximately 1,315° C. A similar relation calculated from Wheaton's data ranged from 0.057 to 0.066.

Many attempts have been made to calculate the equilibrium constant for the reaction



but in most cases little important advancement has been made. The chief difficulty in this connection has been the difficulty in determining the quantities of FeS and MnS existing together in the steel under equilibrium

conditions. Until recently the total sulphur could of course be determined, but the quantities of FeS and MnS could not be differentiated. The electrolytic extraction method offers a possible means for the solution of this problem.

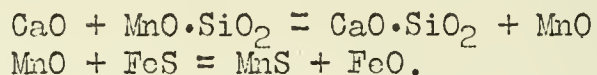
In the basic open-hearth process various reactions may take place which involve sulphur elimination. The Mn-FeS reaction is probably the determining process in the metal bath, but when the MnS particles rise to the slag surface other reactions (principally that between CaO and MnS) occur.

The influence of the basic open-hearth slag and gas phases upon sulphur elimination was studied by Köhler (209). In general, he found that if the basicity of the slag as expressed by $\frac{\text{CaO} - 4\text{P}_2\text{O}_5}{\text{SiO}_2}$ ranges from 2 to 3 the ratio

of sulphur in the slag to sulphur in the metal is more than doubled for the same manganese content. For example, with a basicity of 2 and a manganese content of 0.15 percent, the distribution ratio of sulphur between slag and metal is 4.5. If the basicity is raised to 3 at the same manganese content of the steel the distribution ratio is raised to 7. This effect has been explained in two ways:

1. Increased basicity means an increase in the CaO content, hence the reaction $\text{CaO} + \text{MnS (or FeS)} = \text{CaS} + \text{MnO (or FeO)}$.

2. An increase in CaO content of the slag desulphurizes the steel by the following two reactions:



Manganese Sulphide in Steel. - According to Rohl (313), MnS and FeS form a simple eutectic system with the eutectic at about 6 percent MnS. The melting point of FeS is given at about $1,180^\circ \text{C}$., whereas MnS is shown to melt at $1,620^\circ \text{C}$. The eutectic temperature is slightly lower than the FeS fusion temperature. At the melting point of iron 25 percent FeS is soluble in MnS, and precipitated MnS particles could absorb up to this quantity of the iron sulphide. If sulphide inclusions are ever found in steels uncontaminated with oxides, particularly the silicates, they probably are composed of mixtures of the two. FeS precipitated on cooling from the freezing point of iron could of course be present.

Arnold and Waterhouse (10) determined that steel which was red-short, or cracked and broke on rolling or forging, contained FeS. Microscopic study of this impurity revealed that it was probably soluble in liquid iron but was precipitated in the grain boundaries as envelopes. Additions of manganese gradually changed the appearance of these envelopes to small rounded particles, presumably composed largely of MnS. This concept of the beneficial action of manganese is generally accepted.

Conservation of Manganese in Steel-Making Processes

It has already been pointed out in this section that considerable portions of the manganese introduced into the various steel-making processes are lost. Possibly one of the greatest losses is reported in the basic bessemer process, in which up to 85 percent of this metal is removed by oxidation into the slag phase. Somewhat smaller proportions are dissipated in the other processes, but the losses are always large. Some of the ways in which these losses can be reduced are listed in the following pages.

Use of Liquid Ferro-Alloys

Many suggestions have been made concerning the use of liquid ferromanganese for steel and its advantages over that in which cold alloys are added. It was employed in European plants for many years before it was introduced into this country, the general practice being to melt the alloy in the arc electric furnace and to add the liquid material to the open hearth or bessemer ladle at the time the heat is tapped. Some advantages which have been recorded follow (175):

1. 5 to 30 percent ferromanganese is saved, inasmuch as the liquid alloy is added to the tapping stream and does not come in contact with the slag phase. In the ordinary practice in which the cold alloy is added to the liquid steel in the furnace fairly large lumps must be added some 20 minutes before the heat is tapped, and these lumps are in contact with the slag and therefore are oxidized by that medium before dissolving into the metal phase.
2. The time required to process the steel, particularly in the open-hearth furnace, is shortened because the 20-minute period listed in item 1 is eliminated. If time is not allowed for the cold alloy to dissolve in the metal bath and diffuse throughout, a homogeneous steel will not be obtained.
3. The manganese content of the finished steel is far more uniform with liquid-alloy additions than with solid material.
4. It is unnecessary to raise the temperature of the metal bath before the alloy addition, as is necessary with the addition of cold material. In the manufacture of a number of steels it is advantageous to keep the tapping temperature of the heat as low as possible, and the addition of the liquid alloy is very helpful in this respect.
5. Another advantage in the use of the liquid alloy is that alloy lumps of any size may be used. When cold alloy is added to the furnace it is necessary for the material to be in large lumps so that enough of each lump is well beneath the slag and in the liquid metal so that it will dissolve without great loss. If fine particles of ferromanganese were added the efficiency of the addition would be very low because of

reaction with the slag phase. In the liquid process, however, particles of any size may be premelted in the electric furnace without loss.

6. Sometimes rephosphorization of the metal occurs through reaction of solid ferromanganese with the slag phase, involving deoxidation of the P_2O_5 present in the slag. In the liquid process the alloy does not come in contact with the slag, and because of this no rephosphorization has been observed.

The economy of liquid ferro-alloy additions has been criticized by various investigators, who state that the power requirements for melting ferromanganese or spiegel in the electric furnace are too large to make the process feasible. An itemized list of costs in this practice has been given by Hummel and is found to be approximately \$25 per ton of ferromanganese, of which the power is approximately two-fifths. These figures were obtained on all types of electric furnaces in which the amount of power required per ton of ferromanganese was approximately 700 to 900 kw. h., and in addition to this 80 kilowatts were required per hour for holding the material liquid in the furnace. Unquestionably, these power requirements have since been decreased considerably because of more efficient furnaces, and it is believed that the melting requirement is closer to 500 kw. h. at present. In addition, the price of power per kilowatt-hour has been decreased in this country in the past few years. Other factors that counterbalance the cost of making liquid ferromanganese additions are: (1) the time saved in melting, dissolving, and homogenizing the manganese alloy in the liquid steel; and (2) a decrease in the manganese losses through the elimination of slag reactions.

High Melting, Refining, and Tapping Temperatures

Killing (203) has pointed out that the maintenance of high temperatures throughout the various steel-making processes will increase the amount of residual manganese obtained at the end of the heat. In other words, it is believed that because oxidation of manganese in steel-making processes is exothermic the tendency for oxidation would decrease with increasing temperature. This contention accords with the work of Körber (211) and others on the study of the equilibrium of the reaction $FeO + Mn = MnO + Fe$. The manganese constant given by this author ranges from 260 at $1,530^{\circ}C$. to approximately 100 at $1,800^{\circ}C$., showing that the tendency for MnO to form decreases through this temperature range.

One of the authors knows two open-hearth melters in the same plant; one consistently operates his furnaces at a higher temperature than the other. It has been observed that this melter also has higher residual manganese in his steel before tapping than does the other, although the material charged in each instance is about the same. This fact seems to bear out the contention that high temperatures result in saving of manganese.

Use of Spiegel in Place of Ferromanganese Additions

After the World War, when manganese economy became a major issue of the steel industry, many observations were made concerning the substitution of spiegel for ferromanganese in the manufacture of many different types of steel. Most of those who discussed this point admitted that ferromanganese was necessary for certain grades of steel, but it was also their contention that in the majority of instances spiegel is a very worthy substitute. For example, Hibbard (162) pointed out that spiegel could be used instead of ferromanganese for 70 percent of the steel made in this country. The advantages of ferromanganese listed by that author were: (1) The quantity of material which must be handled is smaller; (2) if ferromanganese is added to the ladle preheating or premelting is unnecessary; (3) the material may be easily broken to any desired size; and (4) ferromanganese has a high ratio of manganese to carbon (approximately 12 to 1) and is therefore useful in making low-carbon steels.

Contrasted with these are various advantages in the use of spiegel. The manganese losses in spiegel additions are lower than in ferromanganese additions, probably because of the function of silicon in the spiegel as a deoxidizer. A spiegel mixture can be used to advantage for recarburization in the open hearth. A liquid mixture composed of spiegel, pig iron, and ferrosilicon may be added to the ladle in such plants where recarburization practice is followed. It has been found that by varying the amounts of these materials in the mixture virtually any analysis for carbon, manganese, and silicon may be obtained. Also, there is no need to add any other deoxidizer, inasmuch as enough silicon is added through the medium of this mixture. When only spiegel is added, it is feasible to charge three fourths of the total amount in the furnace and one fourth in the ladle for high-carbon steels. In low-carbon steels the addition may be divided into halves for furnace and ladle additions.

In answer to a questionnaire distributed by the Iron Age (57) to 70 steel companies in the United States it was found that spiegel is just as satisfactory as ferromanganese for high-carbon steels. During the late war these companies found it necessary to make spiegel additions when ferromanganese was not available or where the price of ferromanganese prohibited its use. Many of these companies are continuing this practice for some types of steel. In the companies questioned, 25 to 75 percent of the ferromanganese additions had been replaced by spiegel. High-manganese pig iron (2 to 3 percent manganese) was also considered in this questionnaire, and it was found to be an advantage by those companies who used it.

Use of high-silicon spiegel. - The substitution of spiegel for ferromanganese was again suggested by Herty (149), who recommended that not only ordinary spiegel could be used for this purpose but also silicospiegels could be used with secondary advantage. This material (152), which varies in manganese-silicon ratio from 4:1 to 8:1, has been found to be a very good deoxidizer of steel, not only because it inhibits the C-FeO reaction but also the products of deoxidation combine to form manganese silicates which, because

of their fusibility and coalescence, are rapidly eliminated from the steel bath. Through the use of this material, at least in the basic open-hearth furnace, the amount of ferromanganese required in conjunction with this type of deoxidizer is 15 to 88 percent less than that required in regular practice. Also, these manganese-silicon alloys are being prepared from some of the domestic manganese ores, and if generally used the import requirements of this country for high-grade manganese ores could be greatly reduced, with accompanying increase in steel quality.

In many American steel companies a standard practice is to reboil the steel with ferrosilicon or silicon pig. The object of this procedure is to hold the steel in the deoxidized condition long enough for a carbon analysis to be reported on the heat before tapping. The steel composition may then be adjusted to meet desired specifications. Considerable information was obtained in connection with this practice at the United States Bureau of Mines (159), and it was found in general that the iron oxide content was essentially the same after reboiling as it was before the silicon addition. In many plant contacts made during this work it was found, however, that the opinion of the practical man was that this not only deoxidized the steel but also removed oxide impurities other than FeO , which were present before deoxidation. In other words, the addition "cleansed" the metal. That this is not true may be seen by reviewing the work mentioned above.

For an alloy to deoxidize steel and at the same time eliminate the deoxidation products, the resultant oxides must form large inclusion particles very soon after the alloy addition. If such large particles are formed they will be eliminated according to Stokes' (317) law or at a rate dependent upon the difference between the density of the oxide particles and the density of the iron and upon their particle size. The particle size is by far the more important factor, inasmuch as Stokes' equation involves the square of the radius. Hence, if an alloy produces large particles immediately after its addition, these globules will be more rapidly eliminated to the slag than will the iron oxide in the slag diffuse back into the metal bath. If the heat were tapped at this time the steel would not only be thoroughly deoxidized (free of FeO) but also would contain little or no oxides resulting from deoxidation reactions.

Manganese-silicon alloys will produce these results, as has been shown previously. The average size of a silica particle resulting from the deoxidation of steel with silicon is approximately 15×10^{-4} centimeters. The average size of particles formed through deoxidation with manganese-silicon alloys of the above mentioned ratios of manganese to silicon is approximately 900×10^{-4} centimeters. If the average radii of these particles were squared and placed in the equation for Stokes' law it would be found that the manganese silicate particle would rise to the slag surface 3,600 times as fast as the pure silica particle. The time required for elimination of the various size particles from liquid steel has been calculated by means of Stokes' law and is shown in table 86:

TABLE 86. - Rate of Rise of Manganese Silicate Particles
in Liquid Steel

Size of inclusion, cm. x 10^{-4}	Maximum rising velocity, inches/minute	Least time required to rise through, minutes	
		30-inch open-hearth bath	12-foot ladle
5	0.061	493.00	2360.00
10	.245	122.00	587.00
50	6.120	4.93	23.60
100	24.450	1.22	5.87
500	612.000	.05	.24

The time required for a given size particle to rise through a 30-inch open-hearth bath (as in a 100-ton open hearth) and also in a 12-foot₄ ladle has been calculated. An inclusion whose diameter is above 100×10^{-4} centimeters would rise from the bottom to the top of a 100-ton open-hearth bath in approximately 1.25 minutes, providing that convection currents in the bath would not force it downward. Under these circumstances any particle above this size would probably be eliminated before samples of the steel could be taken after the alloy addition. That this is true has been proved from practical experience. Manganese-silicon alloys deoxidize steel very rapidly, and very few inclusions of any kind have been found in samples taken 2 or 3 minutes after the alloy addition. Also the steel contains negligible amounts of iron oxide.

Mechanism of deoxidation of steel with manganese-silicon alloys. - The densities of manganese-silicon alloys are somewhat less than those of the liquid steels. Hence, when the alloy is added to the metal bath a portion remains above the surface of the metal. This portion is in contact with the furnace slag, reacting with some of the oxides, particularly the iron oxides. Hence the alloy is added in fairly large lumps to have as much contact as possible with the metal phase, this resulting in greater efficiency of the alloy and its rapid solution in the bath.

Soon after the alloy has gone into solution the manganese and silicon react with iron oxide to form their respective oxides (MnO and SiO_2). The amount of these materials formed depends upon the amount of FeO present in the bath at the time of the addition and the amount of the two deoxidizers added, and also upon the relative rates of the reactions and their equilibrium constants. Both silicon and manganese will tend to establish an equilibrium depending upon these constants, and the balance between the reactants FeO , Mn , and Si and the resultant products MnO and SiO_2 is the relation that determines the formation of large manganese silicate particles. The mechanism of formation of the manganese silicates is dependent upon the slag system $MnO-SiO_2$. It is believed that silica is only slightly soluble in liquid iron, and by far the largest portion of this material will be precipitated as globular particles, whereas manganese oxide has an appreciable solubility in liquid iron. The silica particles would then be in

contact with iron, which is saturated with manganous oxide. At steel-making temperatures silica is saturated with manganous oxide when the resultant silicate contains 40 to 50 percent MnO (158), hence there the MnO tends to be distributed between the liquid iron and the solid silica particles, and it is believed that this distribution is the mechanism which forms large manganese silicate particles. The electrolytic analyses of rapidly solidified steels which had been deoxidized with manganese-silicon alloys indicated that the inclusion particles are usually composed of equal portions of MnO and SiO_2 .

A silicate of this composition has a relatively low fusion temperature, and if any of these particles come in contact with each other it is apparent that they will probably coalesce to form larger ones. Coalescence is a function of probability of contact, convection currents, and surface tension. The surface tension of these particles has been shown previously to be an important property.

Because of their low densities the manganous silicates should and do rise through the metal bath to the slag surface and eventually become a constituent part of the slag. That this mechanism is quite rapid has been already discussed when it was stated that samples taken soon after addition of the alloy were relatively free from both iron oxide and silicate particles, showing not only that the metal has been deoxidized but also that the manganous silicates have been eliminated to the slag phase.

Application of manganese-silicon alloys to steel manufacture. - The amounts of high-silicon spieglers to be added to any steel bath are limited by the following factors: (1) The analysis of the alloy; (2) the specifications of the finished steel; and (3) the minimum amount which can be added and still deoxidize the heat. It is apparent that the amount of alloy added should be kept lower than the manganese specification on the heat. The silicon specification is not so important, inasmuch as only relatively small portions of this element are added through this medium. The carbon analysis is an important factor in the use of this alloy in low-carbon steels but plays a minor role in the manufacture of the higher-carbon steels (above 0.40 percent).

In general, it has been the practice to use a silicospiegel containing approximately 20 percent manganese, 4 to 5 percent silicon, and 4 to 6 percent carbon in the manufacture of high-carbon steels. The silicomanganese alloys, however, are more useful when low-carbon killed steels (0.15 percent) are being prepared. In this case an alloy containing 70 percent manganese, 13 percent silicon, and 2 percent carbon has been used successfully without causing difficulty in meeting the specifications of the product.

The amount of alloy to be added to a given heat is usually expressed in terms of the silicon analysis of the steel. For example, it has been customary to add 0.06 to 0.10 percent silicon as the silicospiegel, etc., as determined by the weight of the charge and the analysis of the alloy. In general, this automatically determines the manganese content, inasmuch

as the ratio of manganese to silicon in the alloy is 4:1 or 5:1; hence the manganese content would range from 0.24 to about 0.50 percent.

Most of the data which have been obtained concerning the benefits resulting from the use of silicospiegels, silicomanganese, etc., have resulted from the application of these alloys to axle-grade steels (155) ranging from 0.40 to 0.50 percent carbon, 0.50 to 0.80 percent of manganese, and 0.10 to 0.20 percent silicon. Two standard practices were in use in the plant in which these heats were made. In one process large additions of ore were made to the furnace to reduce the metalloid contents to low figures as rapidly as possible, followed by the addition of pig iron, ferromanganese, and ferrosilicon in the ladle to deoxidize the steel and obtain the desired specifications. In the other process, the steel was made under less-oxidizing conditions and deoxidized with Alsifer (an alloy of aluminum, silicon, iron, and titanium) in the ladle. Seams and ghost lines were present in steels produced in these processes, and it was the hope of the investigators to remove these undesirable factors.

The seams present in the steels produced by the old process were determined to be caused by the presence of silica, alumina, and aluminum silicate inclusions, whereas the ghost lines were nearly always associated with the presence of sulphides. It was hoped that the use of silicospiegel would eliminate the silicate stringers and the necessity for using aluminum as a deoxidizer and therefore eliminating the alumina particles. The procedure used in making these heats in which the alloy was added was as follows:

The charge usually was 60 percent pig iron and 40 percent scrap, with just enough ore to give a suitable flush slag. The iron oxide content of the slag in virtually all the heats was approximately 20 percent $\text{FeO} + \text{Fe}_2\text{O}_3$. The furnaces were fired with natural gas and coke-oven gas and the slags in nearly every instance were quite fluid. No ore was added to the heats after the carbon content had been reduced to 30 or 40 percent above the specifications of the finished steel, thus preventing the presence of an excess amount of iron oxide at the time of the alloy addition.

The alloy was added in virtually every instance 20 minutes before tapping. The heats averaged 150,000 pounds in weight, and two series of heats were made, in one of which 2,500 to 3,000 pounds of high-silicon spiegel were added, corresponding to a silicon addition of 0.08 to 0.09 percent, and in the second series 3,500 to 3,900 pounds of the alloy was added, corresponding to a silicon addition of 0.10 to 0.11 percent.

In every heat it was necessary to add a small amount of ferromanganese to bring the manganese content within the specifications of the steel. The ferromanganese was added in the furnace, and the time schedule on the deoxidation with high-silicon spiegel and this secondary manganese addition was held as close as possible to 10 minutes between the high-silicon spiegel addition and the ferromanganese addition and 10 minutes between the ferromanganese addition and tapping. The best results were obtained with the higher addition and for this grade of steel has resulted in the recommended

practice. The chief advantage in using the larger addition is that there is less FeO in the metal at the time of tapping than in the other case.

In most of the heats it was essential to add some ferrosilicon in the ladle to meet the desired specification, and because of the lower FeO content much less SiO₂ formed in the heats having the larger addition of alloy.

It was also recommended that fairly high temperatures be maintained toward the end of the heat, and if possible the heat should be held in the ladle to allow for elimination of manganous silicates.

Thickening up the slag immediately after the spiegel addition by means of burnt-lime additions is also an advantage as it prevents diffusion of iron oxide from the slag back into the metal. This can be done by shutting off the furnace gases for a short time, cooling the slag and causing it to become more viscous.

In every heat in which large additions of silicospiegel were made, very few if any seams were found in the finished steel. Inasmuch as it had been previously determined that seams in this particular type of steel were caused by the presence of silicate inclusions, it is obvious that the use of manganese-silicon alloys as a deoxidizer causes elimination of most of the deoxidation products and therefore produces steel which has but few silicate particles present likely to give seams.

Electrolytic analyses were made on all samples and it was found in nearly every instance that the SiO₂ present in the steel was at least 25 percent lower than in the steels prepared by recarburization or Alsifer deoxidation. Other types of steel which have been studied with regard to manganese-silicon alloy deoxidation are low-carbon killed steel, low-carbon rimming steel, and high-carbon seamless tubing steel. The first two grades of material were deoxidized with low-carbon silicomanganese alloys, whereas the latter was deoxidized with a commercial grade of silicospiegel. In every instance decided improvement was obtained when the steel prepared by deoxidation with these alloys was compared to those prepared by standard practices in the various plants.

Probably one of the biggest difficulties encountered in the application of these alloys was concerned with the steel specification itself. Most killed steels being made at present are required to contain a considerable portion of the deoxidizer silicon. In most open-hearth steels the percentage required ranges from 0.1 to 0.2 percent. Electric furnace steels sometimes have a silicon specification as high as 0.3 to 0.35 percent. It is apparent that one of the chief requisites for the successful application of manganese-silicon alloys is that the mechanism of deoxidation as described in these pages should not be disturbed during the deoxidation process.

At the time of tapping a heat which has been deoxidized with the above alloys the steel contains considerable portions of manganese silicates which have not yet been eliminated. If the tapping temperature is high enough the

metal could be held in the ladle so as to allow these particles to rise to the ladle slag and thereby become eliminated. However, many steel companies will not agree to hold a heat in this manner long enough to obtain these desirable results. Instead, it is necessary to add the silicon to meet the above-mentioned specification and to pour the heat as soon as possible. The action of the silicon addition upon the rising manganese silicate particles can be readily imagined. The MnO present is deoxidized with the added silicon, and the finished steel contains considerable portions of SiO_2 particles, not only from the deoxidation reaction but also the original silica present in the manganous silicate.

Steel specification committees should therefore be shown that the two specifications, "clean steel, containing as much as 0.15 percent silicon", are contradictory. This is true if the "clean steel" is being prepared by the use of manganese-silicon alloys. The purpose of the silicon specification is obviously to obtain complete deoxidation. Deoxidation with silicon alone merely involves the removal of FeO and the formation of the inclusion SiO_2 , whereas deoxidation with manganese-silicon alloys involves deoxidation by the reaction of silicon to form SiO_2 and a secondary reaction of manganese to form MnO and subsequent fluxing of these two oxides and their elimination from the steel bath. A "killed" steel may be obtained by either process, but a clean steel can only be obtained by the latter.

Saving of Ferromanganese by Means of Fluorspar in Open Hearth

An interesting suggestion was made by Goldmann (116) which involved the use of fluorspar to save manganese in the open-hearth furnace. This investigator stated that in various heats it was found that manganese was consistently reduced from the slag into the metal by means of additions of this material. The actual mechanism of the saving described by this author is somewhat open to question, but there is a possibility that in the heats studied by Goldmann lumps of lime were floating in the slag at the time of the fluorspar addition. The thinning action of fluorspar on the slag is well known, and the effect of hastening the solution of large lumps of lime in the slag has also been noted. If more lime is taken into solution by this procedure the basicity of the slag is of course increased, and the reduction of manganese from the slag into the metal by this process is more understandable. The basicity of the slag may also be increased by the reaction of fluorspar on the SiO_2 of the slag to form volatile silicon tetrafluoride. This results in an increase in basicity.

Recovery of Manganese from Steel-Making Slags

In 1918 the amount of manganese lost in steel-making slags was estimated to be approximately 114,000 tons (259). Some 10 years later this amount had increased to approximately 279,000 tons, and the increase was attributed not only to the increase in production but also to the practice involving the use of higher-manganese pig iron (151). It was shown that the average MnO content of basic open-hearth slags had increased during this period from 5.5 to approximately 8.5 percent. Many pleas for conservation of manganese to be

found in the literature have pointed out as one of the possibilities the recovery of manganese from basic open-hearth slags.

It is known that during the late war Swedish manufacturers were confronted with similar scarcity of high-grade ores, as in the United States. Also, it has been pointed out that the situation was partly met through their utilization of blast-furnace and electric-furnace slags. At that time the high-manganese pig irons, which were being made in Sweden, were prepared under a blast-furnace slag containing as much as 20 percent manganese. These slags were placed in electric furnaces and reduced by means of coke and ferrosilicon in such a manner that a fair grade of silicomanganese was obtained which could be used for final deoxidation purposes.

Some Swedish bessemer slags were said to contain as much as 50 percent manganese through oxidation of high-manganese low-silicon pig iron in this process. Sixty percent of this manganese was recovered in electric-furnace processes similar to those described above.

In the United States some manufacturers attempted to recover the manganese from the flush slags in the basic open hearth. As is stated on page 265 these slags contain more manganese than the finishing slags from this process because they are run out of the furnace soon after pig iron is added to the open-hearth furnace. During this same process limestone starts to rise from the bottom of the furnace and tends to dilute the slag and thereby decrease the manganese content. The amount of manganese oxide in slags of this type ranges from 10 to 25 percent. One manufacturer who charged slags of this type into the blast furnace recovered as much as 18 percent of the iron and 10 percent of the manganese from the slag. The manganese recovery is not the only advantage which was recognized through this practice. Inasmuch as the basic open-hearth slag contains considerable portions of lime (30 to 50 percent CaO), 50 pounds of lime were taken from the blast-furnace burden for every 100 pounds of open-hearth slag charged. The silica present in this slag was also found to be advantageous, and in general smoother operation of the blast furnace was obtained.

MANGANESE STEELS

The remainder of this paper will be concerned with the influence of manganese upon the properties of solid iron.

In 1882 the researches of Hadfield (136) resulted in the discovery of the peculiar and remarkable properties exhibited by steel when it contained large percentages of manganese. The ordinary properties and nature of the iron were completely changed by such additions of manganese (10 to 15 percent).

Before Hadfield's development of manganese steel there had been just 1 or 2 special manganese products, including the Mushet self-hardening steel, which was rather brittle and useful only for tools.

In 1906 the eminent French metallurgist Leon Guillet (128) proposed the well-known diagram of the constitution of iron-manganese alloys in which all steels above a line connecting 1.4 percent iron and 1.65 percent carbon consist of austenite. Later research by Howe, Campbell, and Hall (144) showed that Guillet did not take into account the presence of cementite in the austenitic steels when the carbon content was above 1 percent. These investigators added the modifying lines XY and WZ of figure 51. Steels whose composition lies between these lines consist of pure austenite when quenched from temperatures slightly above 1,000° C. If carbon is above the line WZ, the equation of which is $C = 1.075 + \frac{0.1}{3} Mn$, free cementite will be found in the steels even after quenching. When carbon falls below the XY line, whose equation is $C = 1.075 - 0.04 Mn$, the steels will contain both martensite and austenite after quenching, therefore the range of useful austenitic manganese steels is considerably narrower than the original Guillet diagram suggests.

As the diagram indicates, there are three principal groups of manganese steels: (1) Pearlitic; (2) martensitic; and (3) austenitic. The various lines converge on 1.65 percent carbon from 5 percent manganese in the pearlitic field and 12 percent manganese in the martensitic field. There is a slight area between each field where there is gradual transition from one form to the other, and both constituents will be found in steels whose compositions fall within such transition ranges.

Neville and Cain (273) have shown that addition of manganese to carbon-free iron affects the properties of iron very slightly. However, as soon as carbon is introduced in conjunction with manganese the influence on the mechanical properties is quite evident, according to the amount of each element present. Each element augments the effect of the other up to certain limits of composition.

Pearlitic Manganese Steels

These steels have been steadily gaining favor in recent years because of certain metallurgical, mechanical, and economical advantages, even though formerly a manganese content greater than was absolutely necessary for the maintenance of sound metal was generally discouraged by users of special steels. The former ill repute of this class of manganese steel may be traced to lack of knowledge concerning the importance of controlling composition, manufacture, pouring, and rolling (171,371).

A manganese content of 1 percent is equivalent to 2 percent nickel in its effect on the physical properties of iron (101). Manganese, acting similarly to nickel, raises the tensile strength and elastic limit of steel considerably without decreasing the toughness. Consequently, by maintaining a comparatively low carbon content and increasing the manganese to the vicinity of 2 percent, the strength will be equivalent to that of a high-carbon steel without the sacrifice of ductility.

The carbon content of the steel should be selected in accordance with the manganese content and the future use of the steel. The steels most commonly found may be divided into several types (102):

Steel type	Percent			
	C	Mn	P	S
1	0.10-0.20	1.20-1.50	0.04 max.	0.075-0.15
2	.15- .25	1.00-1.20	.045 max.	.045 max.
3	.25- .35	1.35-1.75	.05 max.	.075-0.15
4	.30- .40	1.35-1.75	.05 max.	.075-0.15
5	.45- .55	1.10-1.30	.045 max.	.045 max.
S.A.E. 1350	.45- .55	.90-1.20	.04 max.	.055 max.
S.A.E. 1360	.55- .70	.90-1.20	.04 max.	.055 max.

The manganese specifications for the various types depend primarily upon the thermal treatment the steel is to undergo and the mass of material entering into such treatment.

Heat Treatment

Low-manganese steel heat treatment is similar to that of other pearlitic types. Even though manganese does lower the critical temperature somewhat over a straight carbon steel the manganese absorption above the critical is slow, and the heat-treating temperature should not be lowered because of the high manganese content. If the section or casting is heavy and is to be given two heat treatments the first should be somewhat above 900° C. to break up the as-cast structure and completely homogenize the material. The second treatment may be around 850° C. to refine the grain structure.

Uses

Types 1 and 2 (102) may be hardened with a very hard surface at low temperatures and at the same time have a very strong core so that these steels are commonly used in case-hardened parts like cam shafts, roller bearings, gears, etc. Careful heat treatment of these parts after carburization is quite essential to obtain the highest quality product. A double heat treatment after carburizing is recommended; this consists of: (1) Refining and toughening the core by reheating and cooling in carburizing boxes to 840° to 870° C., then quenching in oil; and (2) refining the case by again reheating to 755° to 775° C., then quenching in oil. A final tempering at 150° to 260° C. is desirable when the case is ground, as this prevents grinding checks.

Type 3 and 4 (102) pearlitic steels are used for parts such as heat-treated shafts, spindles, and gears where high strength and toughness are required in conjunction with good machining qualities. Heat treatment for these classes consists of quenching in oil from 830° C. and tempering between 750° and 1,050° C., depending on the combination of hardness, strength, and

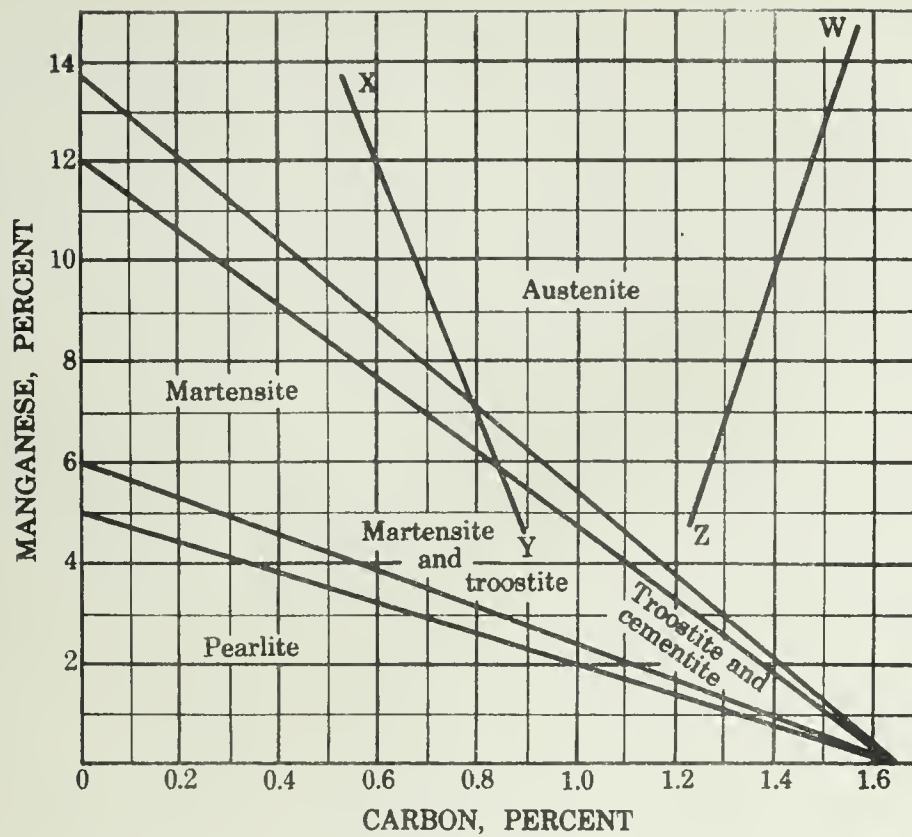


Figure 51.— Modified Guillet diagram summarizing constitution of iron-manganese alloys.

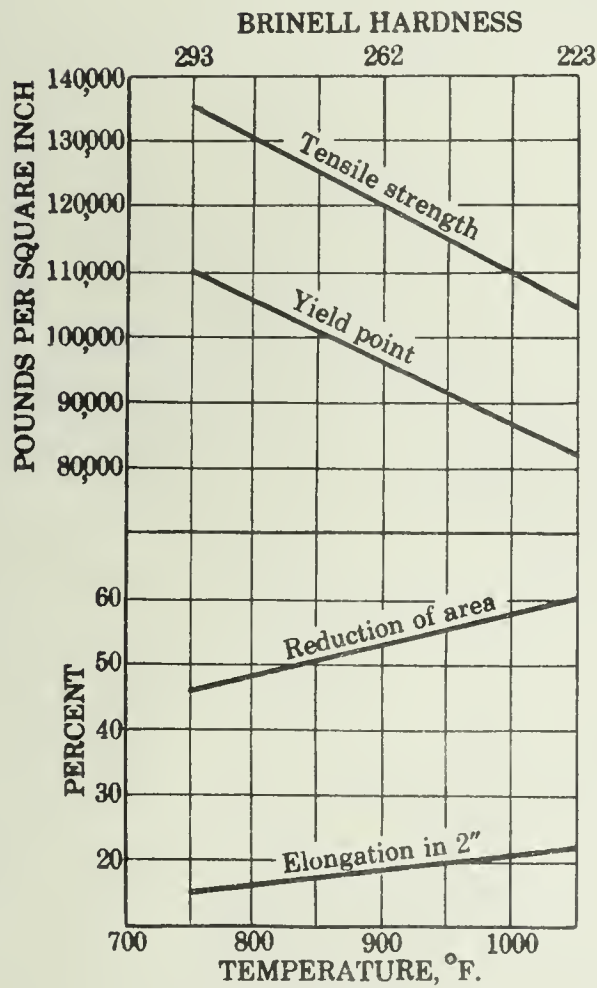


Figure 52.— Physical properties of manganese steel, type 3.

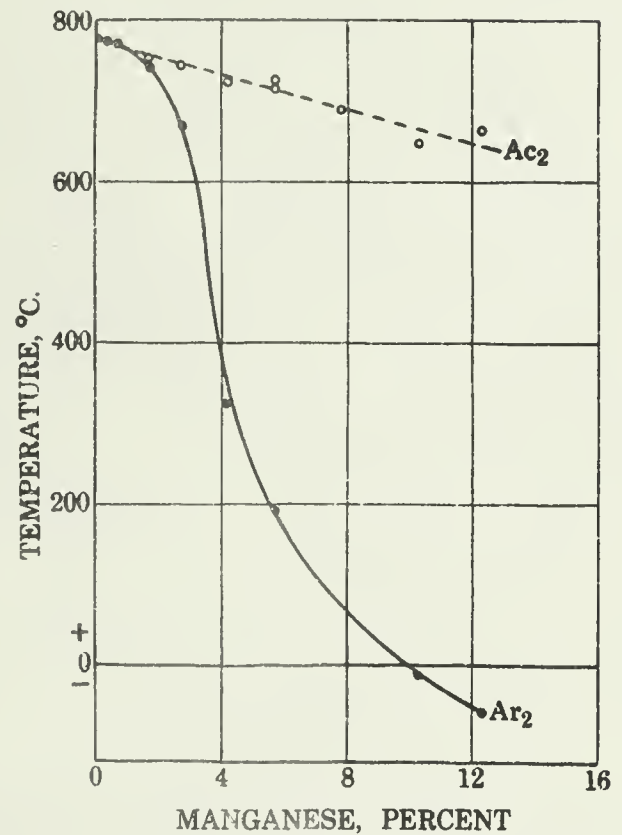


Figure 53.— Magnetic change in manganese steels (Gumlich).

toughness desired in the finished part. Low tempering temperature results in high tensile strength, yield point, and hardness, with low elongation and reduction of area, high tempering temperature resulting in contrary results. Figure 52 shows this influence of tempering temperature on properties.

Type 5 (102) manganese steel is used where physical properties must be of quite a high order, such as clutches, bolts, gears, and shafts. The steels are quenched from 800° to 815° C. and tempered to the desired hardness.

Steels 1350 and 1360 (102) of S.A.E. specification are used for helical springs made from round cold-drawn wire up to three-sixteenths inch in diameter.

In general, pearlitic manganese steels offer no particular difficulty in the manufacture of parts which need to be welded during fabrication. Also, they may be forged in any regular forging operations. These steels may be cold-drawn into wire if properly annealed before drawing.

A steel containing 0.80 percent carbon and 2.00 percent manganese is quite suitable for use as a shear blade in cold-cutting steel bars. It is claimed that this steel gives longer service under similar conditions than any plain-carbon or chromium-vanadium steel of equal carbon content. Masses weighing 500 pounds may be hardened without any difficulty.

The lower-manganese type of pearlitic steels is virtually equivalent to bessemer screw stock in machining qualities, but, as may be expected, machining difficulties increase as carbon and manganese are raised.

Pearlitic manganese steels are appreciably more resistant to sea water and most natural water corrosion than are the ordinary carbon steels. This has resulted in widespread application of a 0.35 percent carbon and 1.35 percent manganese steel in the manufacture of cast-steel anchor chain. It renders good service because of its combinations of corrosion resistance and high strength and ability to withstand suddenly applied stresses.

In the as-rolled and normalized condition these steels are used for steel rails. Standard open-hearth steel rail is measurably improved by lowering the carbon to 0.55 to 0.70 and increasing the manganese to 1.25 to 1.50. If carbon and manganese are both on the high side the rail will be too brittle, while both elements on the low side would give a rail which probably would flow under traffic. By properly balancing the two elements, a long-wearing, tough, generally satisfactory rail is obtained.

Ship and boiler plate, high-strength structural steel, compressed-gas cylinders, and seamless steel tubing, in some of which the manganese approaches 3 percent with correspondingly lower carbon percentages, are further applications. Also, for specialized applications varying percentages of other alloying elements such as nickel, chromium, molybdenum, and silicon often are added to these pearlitic manganese steels. They have not received general application, even in view of their excellent physical properties,

because the manufacturing difficulties encountered in forging and machining operations increase the fabrication cost considerably.

Advantages

These steels are quite applicable in many instances where a superior quality of material is desired and still the expense of a high-priced alloy steel would be prohibitive for the particular use for which the steel is required. The advantages of using pearlitic manganese steels are threefold: (1) Metallurgical; (2) mechanical; and (3) economical.

The metallurgical advantage lies in the lower carburizing temperatures and carburizing time and the uniformity of properties after thermal treatment and decreased danger of deformation on such treatment.

Mechanically these steels show good physical properties and better resistance to abrasion because of their high hardness and toughness.

Economical advantages include 20 to 90 percent speed increase in mechanical operations, less straightening work because of less distortion in heat treatment and 35 to 40 percent longer life of tools.

Martensitic Manganese Steels

As shown on the diagram of figure 51 there is a triangular area covering a considerable range of manganese and carbon contents which is designated martensitic. This region extends from about 5 to 12 percent manganese with low carbon. With increasing carbon content smaller amounts of manganese will produce the same structure. Thus with 0.8 percent carbon, about 3 percent manganese will produce martensite and troostite.

Steels of this class have received no industrial application due to their extreme brittleness, some of them disintegrating to powder under the shock of a hammer. In the cast condition these steels are especially brittle -- in fact, their brittleness is virtually equivalent to that of glass. It is possible to increase the strength somewhat of some martensitic compositions by light hammering or forging, but even after such treatment the product is so brittle that it is commercially useless.

Austenitic Manganese Steels

This important and useful class of steels containing manganese is the result of the classic researches of Hadfield (136) in the early eighties on the influence on the ordinary carbon steels of his day of increasing the manganese content. Previous researches on the effects of manganese additions showed that manganese over 2.5 or 3.0 percent in steel resulted in a metal that was brittle and practically useless. The obvious conclusion followed that there was no sound reason to suppose higher percentages of the same embrittling element would cause any change in this condition. However, Hadfield was unwilling to take the apparently obvious for granted and continued experimenting with increasing additions of manganese.

Hadfield found the same condition of hardness and brittleness in the range of manganese between 2.5 and 7.0 percent at the carbon contents with which he was working. However, above 7.0 percent manganese the steels started to undergo remarkable changes depending on casting temperature and other considerations in physical properties; brittleness and extreme hardness disappeared, strength and ductility increased, and the metal became nonmagnetic. Such remarkable properties, which were diametrically opposed to those of ordinary steel, heralded the appearance of a remarkable discovery and served as the impetus for development of alloys of other elements with carbon as well as the ternary manganese alloy.

The Guillet diagram for alloys of iron, manganese, and carbon, as modified by Hall (144), shows that true austenitic steels will not be obtained by merely having 12 to 14 percent manganese, a corresponding carbon content of 0.6 to 1.2 or 1.3 percent being necessary also. (See p. 293 of this paper.) In fact, the standard austenitic or Hadfield manganese steels contain 11 to 14 percent manganese and 1.0 to 1.3 percent carbon, which falls well within the austenitic range specified by Hall. These steels also contain 0.3 to 0.8 percent silicon and 0.05 to 0.08 percent phosphorus.

Manufacture

Austenitic manganese steel until recently was made in the bessemer converter using a low phosphorus and sulphur cupola metal which was blown soft to between 0.10 and 0.25 percent carbon. To this low-carbon metal molten ferromanganese was added in sufficient amounts to produce a manganese steel containing 11 to 13 percent manganese. The steel was blown quite hot and held just a few minutes for mixing in the ladle before pouring. Such metal was similar in quality to ordinary converter steel in carbide conditions and oxide content.

The disposal of manganese steel scrap presented quite a problem in this process because a metal made from old scrap was rather irregular in composition, and purchasers refused to buy it.

The basic electric furnace (170) offered a solution to the problems of converter practice. First is its absolute power of deoxidation, which is effected under heavy lime slag with a minimum loss of manganese. Also, the metal may be held any length of time without excessive changes in composition, awaiting the results of chemical analysis and thus making regular analyses possible. In addition, the temperature may be controlled closely without resorting to holding the ladle, a practice that might contaminate the metal with slag resulting from corrosive action of the metal on the ladle refractories. Finally, the electric furnace can make satisfactory metal by remelting manganese-steel scrap without any appreciable loss of manganese.

Because of its high manganese content manganese steel attacks the ladle lining, causing considerable slag to form during pouring of the heat. This slag will cause defects in the steel if not carefully held back during

casting. Special tea-spout ladles have been devised for this purpose.

Properties

These steels have the following peculiar characteristics:

1. The alloy may be water-toughened by drastic quenching from red heat. This property, which combines a ductility resembling that of mild steel with a high degree of elasticity, enabling it to withstand shocks that would cause complete failure in ordinary steel, is in exact contrast to carbon steels, which are made comparatively brittle by the same procedure.

2. Even though the steel contains about 86 percent iron, it is non-magnetic.

3. The tensile strength, combined with extraordinary elongation (50 to 70 percent), exceeds that obtainable with the purest iron.

4. The resistance of the steel to abrasion is greater the more severe the service to which it is applied.

The particular properties of these austenitic steels depend upon the treatment given them, this treatment in turn being determined mainly by the use for which the steel is intended. They may be used in the condition as (1) cast, (2) forged, and (3) rolled, the water-toughening treatment being applied in each instance. Austenitic manganese steel greatly resembles cast iron in regard to its fluidity and ability to fill molds of intricate shape, provided full allowance is made for its rather great shrinkage.

Cast-manganese steels, after suitable treatment, possess an extraordinary combination of hardness and toughness which enables them to withstand abrasion quite satisfactorily. This hardness of toughened manganese steel is rather unique and has been termed a tough hardness rather than a flinty hardness. It is a hardness that is peculiar in that it is not practical to attempt to machine or drill it, and yet it may be easily dented by a hammer or marked with a chisel or file. It also has a peculiar flowage when used in crushing machinery, during which the abraded area becomes hardened and still retains the tough austenitic backing under the hardened surface. As this surface gradually wears away subsequent impacts harden another layer of material, and the process continues throughout the life of the casting.

Austenitic manganese steels, like plain steels, are improved in physical properties by hot-working, such as forging or rolling (88,240), even though it is difficult to roll them into anything but simple, plain forms. However, by careful manipulation and control of the steel and proper heat treatment rails, sheets, plates, and even complicated assemblies such as rail crossings may be produced successfully. Usually it is essential for the ingot to be rolled or hammered all over very lightly at first and returned to the heating furnace again before much reduction of size is attempted. The range of temperature within which the steel may be rolled is rather

narrow, because if it is heated too hot it will crumble and if it is too cold it will burst the housings of the rolling mill.

The forging operation must be controlled carefully. A steel which has been well worked or rolled, with proper subsequent heat treatment, will have a tensile strength of 140,000 pounds per square inch and 50 percent elongation in 8 inches compared to the same steel cast and toughened with a strength of 80,000 pounds per square inch and 20 percent elongation. A cold-rolled manganese steel has shown strengths of 250,000 pounds per square inch, but the ductility in this condition is very low.

Manganese steels have a high coefficient of expansion which must be fully considered in designs for castings and in heating steel for the toughening treatment. The castings, which are freer of blowholes than ordinary steel castings, should be so designed that the mass is fairly uniform throughout or that no part is much thicker than any other section because of the liability of the casting being cracked or pulled apart on cooling due to considerable contraction. The poor heat conductivity tends to limit the thickness of sections that may be profitably heat-treated. This limit is usually placed at 4 inches, although slightly thicker sections, in which the presence of internal cracks in the central portion are not ruinous are sometimes heat-treated. If the furnace is much hotter than the piece placed in it for heating the low heat conductivity of the metal retards the passage of the heat to the interior, and the temperature of the skin rises rapidly, and due to the high expansion coefficient the exterior of the piece is expanded so quickly that the interior is likely to become cracked.

A most remarkable property of manganese steel is its lack of magnetic permeability and susceptibility, in spite of the fact that it contains about 85 percent of metallic iron. Moreover, the electrical resistance is about 30 times that of copper and 3 times that of wrought iron. The explanation for this nonmagnetic condition is that the high percentage of manganese contained in the metal prevents the molecular structure from aligning itself into paths of flow for magnetic lines of force.

Influence of Manganese on Magnetic Properties of Steel

The magnetic susceptibility (347) of manganese at room temperature for 99.9 percent manganese is 7.55×10^{-6} . This susceptibility increases abruptly at 310° , the $\alpha - \beta$ transformation, and undergoes another magnetic susceptibility increase at the $\beta - \gamma$ change.

The addition of nonmagnetic manganese to an iron-carbon alloy or steel causes gradual decrease in magnetic properties, until at a manganese content of 13 percent ordinary water-toughened austenitic steel is practically nonmagnetic. Steels with manganese contents between 2 and 13 percent show a decided hysteresis effect in the magnetic transformation on heating and cooling. This hysteresis is rather pronounced between the Ac_2 and Ar_2 transformation points. The magnetic properties of manganese steels as indicated

by figure 53 show that a 10 percent manganese content (carbon, 0.1 to 0.2 percent) requires cooling below 0°C . before magnetism is recovered if the alloy has previously been heated above the Ac_2 point.

Hadfield and Hopkinson (141) could find no simple relation between the proportion of manganese and the reduction in magnetism in iron-manganese alloys. They did find that the lowering of magnetism by manganese is affected by small amounts of carbon, but they also noticed other anomalies attributed to the temperature to which the samples were subjected during forging and rolling.

Table 87 illustrates the effect of carbon and manganese on magnetic saturation.

TABLE 87. - Effect of Carbon and Manganese on Magnetic Saturation

Sample no.	Analysis, percent			Relative magnetic saturation, percent (pure Fe = 100)
	C.	Si.	Mn	
37	0.08	0.130	3.50	90.9
38	.15	.370	5.40	93.6
39	.16	.630	10.08	44.5
40	.15	-	15.27	4.5
41	.41	.070	2.23	95.0
42	.78	-	3.81	54.3
43	.36	.100	4.68	91.6
44	1.00	-	7.00	74.0
45	1.66	-	11.53	42.0
45	1.66	-	11.53	41.5
46	1.66	-	11.53	2.0
47	1.23	-	12.64	-
48	.26	-	13.00	15.0
49	1.50	.14	15.22	8.5
50	1.54	-	18.50	0.0

In No. 38, 5.40 percent manganese acts like an inactive diluent, while in No. 37, with low carbon, the percentage reduction is three times the percent manganese. An additional 5 percent manganese caused a 50 percent reduction in magnetism.

Hadfield and Hopkinson found that by heating the nonmagnetic austenitic alloy to 520°C . for a long period the steel has an amount of magnetism corresponding to about two thirds that of pure iron. The material could be rendered magnetic by heating below 520°C ., but the change was much slower. Heating above 700°C . rendered the material only slightly magnetic.

On the destruction of this magnetic quality by heat treatment, any temperature of heating below 550° caused no diminution of the magnetism, but it is diminished by heating at 640° C. or above. A few minutes heating at 750° C. virtually destroys the magnetism, if the steel is cooled either by quenching or by air.

Hardness

Manganese steels owe their great wear-resisting qualities to the conditions which Hadfield (138) expressed as a "peculiar hardness combined with a special kind of softness", which in reality is a combination of hardness and toughness or ductility, producing certain results under definite conditions of deformation or abrasion. Castings or forgings (284) of these steels are too hard and brittle for practical use, due to deposition of a double carbide of manganese and iron in an acicular form. Reheating to bright-red heat and quenching in water reconverts this precipitated carbide into the crystalline or so-called gamma solid-solution state, with a hardness of slightly over 200 Brinell and the ductility which renders it so valuable. The toughness it possesses in greater measure than any other metal is due to its very closely knit, cohesive grain structure, the result of proper water toughening. When this steel is subjected to wear either abrasive or incisive the structure knits together even more firmly than in its unworn state, enabling it to resist abrasion or cutting.

This remarkable abrasion resistance is explained by the somewhat unstable crystalline condition of the metal. Any steel, when heated above its critical temperature, passes into a nonmagnetic, gamma iron, austenitic state. Most alloying elements, including carbon up to 1.7 percent, are soluble in gamma iron. Even severely quenched steel cannot be retained in the gamma condition unless some obstructive element is present. Manganese content of 10 to 14 percent with 1.0 to 1.5 percent carbon acts as such an obstructive element. When the steel is heated to about $1,040^{\circ}$ C. the same nonmagnetic austenitic structure is obtained. The high manganese content obstructs the gamma to alpha change and the precipitation of iron carbide, and by rapid quenching the mass is maintained in an austenitic condition but in a somewhat metastable state.

This quenched, strong, tough, austenitic steel is so unstable that cold work or even impact will cause some gamma to alpha transformation with precipitation of carbon. This unique property of surface hardening under mechanical deformation makes the steel valuable to industry. The hardening from about 200 Brinell to as high as 550 at point of abrasion or impact is the secret of the durability of the austenitic manganese steels.

The best wear-resisting qualities are obtained under the hardest working conditions. Its hardness during machining is due to the deformation produced during the cutting action and pressure of the tool, which in its turn causes increase of hardness and causes the machining process to be impractical.

The same conditions explain the difficulty encountered in drawing manganese steel into wire, the passage through the dies causing the hardness following mechanical deformation.

Heat Treatment

The unique and useful properties of austenitic manganese steels depend even more on the heat treatment and cooling to which they are subjected to develop the great toughness and ductility than the highly important consideration of composition of the steel. These steels show the distinctive characteristic that quenching and annealing produce effects opposite to those obtained in most other steels, quenching making the alloy tough, ductile, and non-magnetic, while annealing makes it hard, brittle, and partly magnetic.

To obtain maximum toughness and ductility, manganese steels are quenched in water from a high temperature, but the rate of heating, maximum temperature, and rate of cooling may be varied considerably without changing the results materially. Thus while in ordinary steels the heat treatment is guided more or less by transformation temperatures, high-manganese steels have no critical points, so that the only criterion of the proper temperature is the relation of the carbide to the physical properties. The absorption, with or without the precipitation of such carbides, is the underlying basis for heat treatment adjustment.

The thermal treatment for these steels involves two distinct factors: (1) The change of grain size; and (2) the relationship of austenite and carbide, with or without the presence of martensite. Martensite may be omitted, since commercial manganese steels do not naturally contain it.

The necessity for the first consideration is obvious. Manganese steel as cast is fundamentally austenitic, the crystals often being excessively large and in many instances forming a weak, columnar structure. Forged manganese steels may have fine grain structure and still be weak and brittle. Annealing would cause grain refinement but is illogical, because it has an injurious result on the steel -- the formation of the maximum amount of carbide, an extremely hard, brittle manganitic cementite rejected by the austenite, which forms a weak membrane around the austenite grains as spines and needles.

Heating to high temperature causes the membrane of double carbide of manganese and iron, or carbide segregation, produced by cooling after casting to be taken into solution gradually by the austenite, and by rapid cooling this carbide is more or less prevented from reprecipitation. The temperature of heating must be high (about 1,050° C.) and the cooling drastic, such as water quench, to give the most ductile steel.

When castings undergo some heating after having received the water-toughening treatment carbide precipitation occurs if the temperature goes over 390° C. impairing its ductility and wear-resistance properties. Temperatures below 390° C. may be accommodated without losing any of the remarkable ductile properties. If heating has been carried near the danger point it is advisable to take it to 1,050° C. and re-quench.

This embrittling effect due to destruction of austenite by precipitation of carbides when the steel is heated above 390° C. proves valuable in the recovery of large pieces of manganese-steel scrap. It is merely necessary to heat to a slight red and cool slowly in the furnace; then the section can be broken as easily as cast iron by means of a drop ball.

Uses

The extreme difficulty encountered in machining the austenitic manganese steels has limited their use considerably. The steel is usually applied in such conditions where the castings or forgings can be used without further machining. Sometimes a final finish is given by grinding.

Manganese steel receives wide application in uses requiring resistance to abrasion under slow speeds of impact, such as in rolls, crushers, and car wheels running at slow speeds (on mine cars, for example). Moreover, for railway-track frogs, switches, and curved rails the steel is used extensively. Manganese-steel rails abrade more slowly on sharp curves than bessemer or open-hearth steel rails.

The steel is an ideal material for burglar-proof vaults and safes. In carbon or chromium steels the burglar's blow pipe will draw the temper locally by simple heating so that as soon as it is cooled the metal can be drilled and the dynamite charge introduced. Such treatment does not soften manganese steel but makes it harder. Also, the steel is too strong to be broken by dynamite and nitroglycerin.

Because of its nonmagnetic property the steel is employed for cover plates of lifting magnets handling heavy iron and steel articles where it is subjected to hard blows of the pieces attracted to the magnet. It offers virtually no obstruction to the passage of the magnetic attraction. The steel is also used around the structure of the compasses on ships because it does not affect the compass needle. Railway switches and diamonds are covered with a layer of manganese steel by arc-welding a 13.8 percent manganese and 1.23 percent carbon steel on the rail. After being worn down it can be refilled. Adherence is improved by drilling holes at an angle into the rail.

This steel is most applicable for use in equipment employed in the mining, milling, and treatment of ores and other mineral products, quarrying and rock dressing, digging and dredging, or parts of machines exposed to gritty wear. Mere hardness is not the only quality desired for these purposes, for in that instance chilled iron would be the logical material, whereas manganese steel, which is not as hard as chilled iron but has the added property of being as tough as mild steel, will wear as much as 4 to 7 times as long in this type of service.

It is common belief that austenitic manganese steel must be as low as possible in carbon to attain maximum ductility, the manganese : carbon ratio being as high as 10.00 as can be obtained. However, a comparison by Barton (15) of three series of compositions which were above the minimum ratio number had quite different ductilities.

	Average content, percent		Ratio, Mn : C
	C	Mn	
Series I.	0.96	10.94	11.40
Series II	1.05	11.61	11.00
Series III.	1.22	12.74	10.40

Series III bent cold on the standard bar test 180° without cracking. Series II bent only about 120° before incipient cracking. Series I began cracking at 90°. Therefore, although the steels all fall within the chemical specifications, a steel of lower quality than might be desired for some purposes sometimes results. Therefore, for parts where extreme ductility and resistance to shock are desired, a manganese content between 12.50 and 14.00 percent should be specified.

A special application (137) which resulted in the saving of many lives was found during the World War in which, due to its high tenacity and ductility, it was used in body shields and helmets. This high tenacity, combined with extraordinary elongation, enabled shields of manganese steel to resist penetration of bullets by bulging rather than penetration, the material taking up the impact by indenting gradually and thus offering a larger surface of resistance.

Due to its great toughness and tensile strength, which practically eliminate breakage, rolled manganese-steel plate is used for dipper doors on shovels for crushed stone-quarries. The pieces handled are large and heavy, causing considerable wear and breakage when any other steel is used on the dipper doors. Woven screens of manganese-steel bars give a life many times that of screens made of soft ordinary steel. Coke acts as though it contained many minute diamonds, which rapidly wear away the metal on which it impinges.

If the cost, due to the hardness developed during the process of drawing into wire, were not so great, manganese steel would be an excellent resistance material in the form of wire because of its high electrical resistance, its specific resistance ranging from 65 to 75 microhms per cubic centimeter. It does not wear so well when used with sharp easily friable materials, because such use does not harden the surface enough.

Because of its low yield point, manganese steel does not give satisfaction in many lines for which it would otherwise be fitted. If the pressure at the point of wear passes the flowing point the results are unsatisfactory, though even then they may be better than those from the use of other material.

MANGANESE IN CAST IRON

Cast iron is an alloy of iron, carbon, and silicon, in which the carbon is so high -- over 2.2 percent -- that the alloy is not malleable as cast at any temperature. Bolton (29) has classified cast iron as follows:

1. Pig iron.
2. White cast iron.
 - (a) Chilled iron.
 - (b) Malleable iron.
 - (1) American or black heart.
 - (2) European or white heart.
3. Gray cast iron.
 - (a) Ordinary gray iron.
 - (b) Semisteel.
4. Special irons.
 - (a) Pearlitic iron.
 - (b) Mottled iron.
 - (c) Alloy iron.

Pig Iron

Manganese tends to increase the total carbon content of pig iron by forming stable carbides, provided the manganese content is higher than that necessary to combine with the sulphur to form MnS. The excess manganese forms a double carbide, $(\text{FeMn})_3\text{C}$, and thereby increases the proportion of carbon in the combined form and also results in higher total carbon in the pig iron. Sulphur has an equivalent effect of increasing combined carbon content, but when present with manganese the two elements combine to form MnS and thereby neutralize each other as far as increasing combined carbon is concerned. Consequently, when manganese is high in pig iron an increase in sulphur will decrease the amount of $(\text{FeMn})_3\text{C}$ and thereby decrease the tendency of manganese to raise combined carbon as well as total carbon.

With the exception of direct metal used as a hot addition to a steel-making charge, blast-furnace iron is cast into pigs and later either is remelted and cast, producing gray and white cast irons, or is refined in the steel-making process.

White Cast Iron

White cast iron contains almost all of its carbon in the cementitic or combined form. The iron may be obtained in two classes -- chilled or malleable -- depending on the casting procedure or treatment given after casting. Casting the iron against metal chills produces the chilled variety of cast iron.

Low manganese content will have little appreciable effect, silicon and sulphur having greater influence, on the chill cast iron produced because it merely acts as a deoxidizer and desulphurizer, cleansing the metal from impurities, and greatly increasing the strength and ductility of the casting without injuring the chilled section. When alloyed in higher percentages -- 2 percent or more -- in the presence of silicon, the manganese exerts a stabilizing effect on the eutectic cementite, producing a hardening and chilling effect of a coarse, crystalline nature.

Manganese increases the shrinkage of white-iron castings because of the tendency for manganese to retain carbon in the combined form. Additions of enough silicon to cause graphitization of some of the cementite decrease the shrinkage considerably. The hardness of white cast iron is increased by manganese, which hardens both by increasing the combined carbon and also by forming an iron-manganese carbide, $(FeMn)_3C$, which is quite hard.

The malleable variety of white cast iron is further subdivided into two classes, depending on the treatment after casting. The European treatment gives the so-called "white-heart" malleable castings, while the American process is modified to produce the "black-heart" castings. The metal must be of such composition that it is just on the line between the precipitation of graphite and its retention in the form of cementite and dissolved carbon. In all instances the majority of the carbon must be dissolved or combined. The castings are reheated to 625° to 875° C. and maintained at this temperature for 60 hours, which precipitates graphite as temper carbon or temper graphite.

White-heart castings are made by the same method, but the annealing is prolonged until the pieces are practically decarburized to the core as well as on the surface.

The manganese content in malleable cast iron must be confined within certain maximum amounts. Manganese must be twice the sulphur content; even three times is preferable, except when the sulphur content is 0.3 percent or more. Some authorities set the limit for manganese at 0.3 percent. However, 0.5 percent of manganese aids in decreasing checking; also it protects silicon from oxidation both during melting and during annealing, and on this account aids in completing precipitation of the temper carbon in the malleable castings. The manganese should never exceed 0.6 percent in this grade of cast iron, as it makes the casting hard and difficult to machine or anneal.

Gray Cast Iron

Ordinary gray cast iron is so designated because of its characteristic gray fracture. Its composition is such that none of the carbon is retained as free cementite but is present as a constituent of pearlite or as graphitic carbon flakes. Proper decomposition of the combined carbon in massive cementite is accomplished by adjustment of the silicon content. Just enough should be present to accomplish graphitization of free cementite but not affect the pearlite.

As gray iron depends on proper graphitization, manganese should be maintained within certain limits which will not have too great a carbide-stabilizing effect on the graphitization influence of silicon. Manganese has been considered to assume several forms, such as manganese carbide associated with cementite in the pearlite of the cast iron and manganese sulphide which occurs as small rounded particles throughout the metal. Manganese content in ordinary commercial ranges has very little influence on the properties of gray iron.

The slight influence of manganese on gray cast irons is an increase in tensile and transverse strengths up to about 1 percent manganese, above which the strength again decreases. Hardness also increases with manganese content. However, when the manganese is low the hardness remains virtually the same because of greater graphite separation, as small amounts of manganese up to 0.3 percent increase the graphite separation in gray castings containing about 1.5 percent silicon. Further increase of manganese to 2.5 percent does not affect the separation of graphite.

The effect of increasing manganese content on shrinkage is not so great in the gray iron as in the white cast-iron product because of the formation of some secondary graphite during cooling.

The semisteel variety of gray cast iron is merely a gray iron which has been diluted with some steel scrap, thereby lowering the content of silicon, phosphorus, and total carbon. These lowered compositions result in the graphite being distributed in a fine state of division in the purest possible form of pearlite. Fairly high manganese, combined with low carbon and silicon, is responsible for attainment of the desired fine-grained structure.

Special Cast Irons

The special irons, such as pearlitic irons, high-strength irons, and alloy cast irons, are not greatly influenced by manganese content, their particular properties being obtained by proper casting, adjustment of carbon content, or addition of the alloying elements nickel, chromium, vanadium, and molybdenum. The manganese content of these special irons is merely maintained within the ordinary commercial range of 0.35 to 0.90 percent.

General Effect of Manganese on Cast Iron

Manganese lowers both the temperature of primary solidification and the eutectic temperature of the iron-carbon system. The latter temperature is gradually lowered to $1,120^{\circ}$ at about 13 percent of manganese. Above this amount the solid solubility increases and the melting point is gradually raised to $1,260^{\circ}$ C. at 80 percent of manganese.

The eutectoid temperature is not changed very much by manganese. However, by increasing the hysteresis the temperature at which inversion occurs is greatly affected. Each percent of manganese lowers the A_{r1} transformation point of iron-carbon by 20° to 25° C. (1). The pearlite becomes finer up to about 5 percent manganese. Above this amount the gamma solid solution remains undecomposed down to room temperature. The effect of manganese on cast iron containing silicon, as it ordinarily does, is somewhat modified by the silicon. It has been shown that 2.50 percent silicon is enough to precipitate graphite in the presence of 16 to 18 percent manganese. Also, manganese does not lower the freezing point of the iron, but the A_{r1} point is lowered even more rapidly than in the low-silicon or white iron series. The pearlite point disappears at about 4 percent manganese and is an explanation as to why high manganese produces hard, white cast iron. Moreover, the gamma solid solution

appears before graphite disappears, tending to make iron white.

Manganese exerts a detrimental effect on the magnetic properties of cast iron and consequently should be as low as possible in castings where high induction and high permeability are desired. Also the remanent magnetism and coercive force decreases so that cast irons with manganese are not suitable materials for permanent magnets.

Manganese aids in counteracting the effect of phosphorus, which tends to produce large crystals, thus increasing the liability of the metal to check. By reducing the crystal size with manganese this checking tendency is decreased. More than 0.4 percent of manganese must be present in cast irons to cause any appreciable effect on mechanical properties. Increasing manganese results in some increase in tensile strength.

There is no influence on elongation by manganese up to 1 percent content (236), but higher percentages than 1 percent cause a decrease in elongation. By long annealing the manganese content may be somewhat higher than 1 percent without affecting the elongation adversely.

The retardation in cooling corresponding to the transformation of gamma into alpha iron sinks with increasing manganese content to lower temperatures with increasing manganese percentages until at about 8 percent manganese the thermal irregularities disappear entirely.

An expansion takes place corresponding to the gamma-alpha transformation which exerts an effect opposite to that of shrinkage. For this reason the low-manganese irons show a distinct expansion in the transformation region. At higher manganese percentages the transformations require longer periods and the temperatures at which they occur are lower.

Consequently, the manganese in cast irons must be definitely controlled within certain percentages, depending on the grade or purpose for which it is used.

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PART V.

CHAPTER 10. BIBLIOGRAPHY

CHAPTER 11. INDEXES, BY MABEL E. WINSLOW



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MANGANESE: ITS OCCURRENCE, MILLING,
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PART V

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UNITED STATES BUREAU OF MINES
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INFORMATION CIRCULAR

LEASING SYSTEM AS APPLIED TO METAL MINING



BY

W. O. VANDERBURG

INFORMATION CIRCULAR
UNITED STATES BUREAU OF MINES

LEASING SYSTEM AS APPLIED TO METAL MINING¹

By W. O. Vanderburg²

INTRODUCTION

The leasing or tribute system has been an important factor in the development of our metal-mining industry, particularly in the gold and silver mines of the Western States. It has received its widest application in the early history of many camps and in the last stages of many mines. The system is especially adapted to mining where the ores occur in narrow veins or are erratically distributed in small bunches or pockets so that selective mining, hand sorting, screening, or hand cobbing is required to produce shipping or milling-grade ore. The leasing system is also applicable to the development of prospects where the owners are unable or unwilling to assume the risks of searching for ore.

The virtue of the system lies in the opportunity it affords miners with little capital to get started; at the same time, it gives owners an opportunity to have their property developed at small cost.

In the following pages leasing is defined; a brief history of the system is given; its advantages and disadvantages discussed; and as an aid in drawing up a lease the terms ordinarily included are set forth. In regard to the terms of a lease the statements should be taken as illustrative rather than applying exactly to average operating conditions. In the final analysis, just how the details of the leasing system can be applied to a property is a problem which must be worked out in each case.

DEFINITION OF LEASING

The term "lease" as ordinarily used in metal mining is a written or verbal contract for the exclusive possession and use of real property with specified restrictions, for a definite time, and for a stated consideration called rent, royalty, or tribute, payable at fixed times with money, in part of the product, or an agreed percentage of the proceeds derived from the sale of the mineral. The last-mentioned method of payment is the one commonly employed in the United States. The party taking the lease is variously called a grantee, lessee, tributor, adventurer, or chlorider.

¹ The Bureau of Mines will welcome reprinting of this paper, providing the following footnote acknowledgment is used: "Reprinted from Bureau of Mines Information Circular 6774."

² Associate mining engineer, U. S. Bureau of Mines.

According to Lindley³ mineral in place is land and an estate in such mineral may exist as an entity entirely separate from the overlying surface. Removal of minerals from the land is an act which constitutes permanent destruction of real estate. Unlike a business lease the relation of landlord and tenant in a mining lease does not contemplate destruction of the estate, and such destruction cannot be called use. Therefore, while the contract is in name a lease it amounts to a sale and the so-called royalty is purchase money for real estate.

The California courts have held that a contract giving the right to work a mine for a certain time (the gross profit to be divided between the parties) is not a lease; that such contract does not create the relation of landlord and tenant but fixes a rule of compensation for services rendered, and that in all its essentials it is a contract of labor to be performed and paid for by a share of the profits.⁴

According to this interpretation leasing may be defined as a pro-rata method of paying miners' wages. It differs from the contract system of paying wages used at some mines, by making the miner a sharer in the risk as well as in the profit.

Although it may be difficult to determine the correct legal name for the contract employed, this is less important from a practical viewpoint than ascertaining from the contract the respective rights of the contracting parties.

In some States a verbal lease on mining property is as binding as a written one, which is contrary to the general law that any transfer in the interest of land shall be void unless in writing. However, it is best for the protection of both parties that a mining lease be in writing and properly recorded.

Lessees are not always individuals but may consist of a number of men associated as a company for the specific purpose of leasing.

Leases are not confined to mines proper and prospects but may also include mine dumps, mill tailings, or mills, which contain recoverable minerals.

BOND AND LEASE

The term "bond and lease" as applied to mining is commonly used to cover a contract whereby the owner of the property agrees to sell it at a stated price and deliver a valid deed for it before the lease expires. A bond and lease provides for the possession and working of the property by the lessee, usually with the right to appropriate the mineral, with payment of stipulated

³ Lindley, Curtis H., Lindley on Mines: 3d ed., vol. 3. 1914, pp. 2135-2136.

⁴ Mining and Scientific Press, vol. 89, 1904, p. 3.

royalty on the ore extracted. It usually gives the lessee ample time in which to determine the merits of the property, and in the event of disappointment in its worth, he may terminate the contract. Since the value of mineral land with little or no ore in sight is extremely difficult to determine, a bond and lease on a property furnishes the lessee an incentive to do exploration and development work to demonstrate the value of the property before purchase.

HISTORICAL DEVELOPMENT OF THE LEASING SYSTEM

The evolution of the leasing system is historically interesting, since it has played a prominent part in the development of mining from remote times.

In ancient Greece the mines were considered the property of the state and were worked by lessees who paid tribute to the state either in the form of initial purchase or annual rent. This conception of mineral-land ownership was inherited from older civilizations.

Under Roman law, likewise, the mines obtained by conquest belonged to the state, which leased them to operators or worked them directly with slave labor.

Before the 13th century the miner or landlord had few rights to minerals, but with the rise of individualism about this time a number of charters appeared in Europe and England which confirmed to mining communities certain privileges but all reserved the regalian right whereby some minerals, particularly gold and silver, could be mined only on payment of royalty to the crown.

In England the mining communities of Cornwall and Devon were in the 13th century chartered into corporations called the "Stannaries", with broad powers but with the reservation to the crown of some sort of royalty. Eventually the miners and landlords obtained greater rights at the expense of the crown, and gradually the leasing system under private control came into existence.

The basic features of the leasing system in the United States probably were derived from England. The policy of leasing mineral lands in the United States by the Federal Government was inaugurated under Act of Congress March 3, 1807, wherein lead-mineral lands in the Indiana territory (Louisiana Purchase) were removed from sale, with the provision that they should be leased on an annual rental. The leases were under the supervision of the War Department. Royalty of one eighth to one tenth payable in lead or cash was demanded. The act provoked opposition, so that President Polk, in his first message to Congress, December 1845, recommended its repeal. This first attempt at leasing mineral lands by the Federal Government was abolished in 1846.

Although the leasing of mineral lands by the Government proved unsatisfactory in the early days the system, under corporate or individual control, gave a tremendous impetus to metal mining in the West. The leasing system was introduced into the mining camps of the West in 1859 in the Gregory district, Gilpin County, Colo. It spread rapidly to other Western States, and attained its greatest development in Colorado and Nevada.

Among the districts where leasing has been a prominent feature in the operation of mines may be mentioned Cripple Creek, Aspen, Leadville, in Colorado; and Eureka, Tonopah, Goldfield, in Nevada. Numerous other mining camps could be cited where leasing has been employed at some time in their history, but the above will indicate that the system has been an important link in the chain which forms the stage-by-stage establishment of commercial mining risks, and also a noteworthy feature in working old mines where individual miners take chances of making money under conditions which do not appeal to companies.

ADVANTAGES AND DISADVANTAGES OF LEASING SYSTEM

The leasing system is particularly applicable in working old mines where the ore deposits are small and occur erratically and the finding of ore is the principal difficulty; or where the ore occurs in narrow veins, or bunches, which must be mined carefully and concentrated by hand to produce a shipping or milling grade product. In these cases the leasing system offers advantages compared to company operations on a days-pay basis.

A lessee has a sense of personal possession in part of the mine, and his primary aim is to make his lease as productive as possible. This sense of ownership provides a strong mental stimulus and brings forth intensified effort, since the lessee realizes that his success depends on his own skill and labor. By financial participation in the results of personal effort the lessee is also willing to take financial risks that the company may not be willing to advise, in the hope of making a strike, and by so doing he is occasionally rewarded with success.

The lessee generally works with a minimum of capital and consequently divests his working costs of every unessential item. He is primarily interested in profits with the least possible expense, and in this he is generally more successful than some companies which install costly plant equipment before enough ore has been found to justify the expense.

An important feature of the leasing system from the viewpoint of community prosperity is due to the fact that the lessee is frequently "grubstaked" by local residents, and any success attained leads the way to further effort.

Formerly it was considered at least a tacit admission that the end of a mine was in sight when it was thrown open to lessees. In recent years, however, some large mining companies have employed leasing as an adjunct to their regular operations. As an example, the El Tigre mine in Sonora, Mexico, producing gold and silver, was forced to make drastic reductions in wages and personnel in 1931, due to falling silver prices. Continued operation was made possible by resorting to the leasing system.⁵

Where leasing is conducted under the general supervision of the management and where care is exercised in the selection of well-qualified men the system

⁵ Wasley, W. A., The Mexican Gambusino in El Tigre: Min. and Met. April 1933, p. 187.

works out more satisfactorily than the old method when the lessees were left to their own devices. In the latter event only the best ore was gouged out, the workings left in poor shape for future operations, and any low-grade ore left behind could be mined only at considerable expense.

In many mines there is a time when at least part of the territory can be worked to better advantage by lessees than by any other method. Two principal reasons are: First, the lack of efficiency and the expense of supervision for small operations distant from the larger stopes; and second the close attention required in hunting and mining small bunches or stringers of ore frequently overlooked or left behind intentionally in the rapid mining of large ore bodies.

An advantage of the leasing system from the property owner's viewpoint is that the lessee is responsible to the owner for damage to the mine, while under the days-pay system the owner is responsible to his employees. Also, the mine owner is not liable for liens when his property is worked by lessees.

The leasing system has been condemned because the contention is made that development work is apt to be done grudgingly and with little attention to subsequent utility. To this objection it may be stated that if adequate supervision is maintained over the lessees' work there is little reason why it will not come up to company standard.

The argument is also made that the lessee can afford to mine only high-grade ore, leaving the lower-grade material that by itself cannot be mined at a profit. This objection can be remedied by proper adjustment of royalty payments on the lower-grade ores.

TERMS OF A LEASE

There is no standard form of a lease suitable for all mines. The terms of a lease are varied to suit the conditions at each mine or the ideas of the owner and lessee. The elements of a lease may be summarized as follows:

- 1 - Date; names of contracting parties.
- 2 - Duration of lease; date of beginning, date of expiration, and whether or not subject to renewal.
- 3 - Description of lease; location and name of property; description of leased ground by definite boundaries.
- 4 - Royalty; amount of royalties and how calculated; when and how paid.
- 5 - Manner of working lease.
 - a - Minimum number of working shifts required per month by lessor.
 - b - What constitutes abandonment and forfeiture of lease.
 - c - Ownership of plant and improvements made by lessee when lease is terminated.
 - d - Safety of premises.
 - e - Disposal of waste.
 - f - Preservation of adjacent parts of mine, keeping levels clear, timbering and repairs, stoping near shaft.

g - Minimum dimensions of underground workings, namely, shafts, winzes and raises, drifts and crosscuts.

h - Underhand stoping, whether or not permitted.

i - Work subject to inspection and direction of company agent.

j - Where ore is to be milled or smelted.

k - Bills incurred by lessee to be settled promptly and vouchers to be shown lessor on demand.

l - Subleasing, whether or not permitted.

m - Ownership of dumps when lease is surrendered.

n - Weighing and sampling ore.

o - Liability for damages or indebtedness.

p - Charges for compressed air, hoisting, transportation, blacksmithing, or other equipment and services furnished by lessor.

The principal points of difference in lease contracts are duration of lease, size of leased ground, and royalties.

The duration of the lease is nearly always specified, and it may range from 6 months to several years. Leases covering precious metal mines and prospects usually are given for 1 year, sometimes with the privilege of renewal for another year. Occasionally a lease may be of indefinite duration, but the lessee's operations are confined to a restricted block of ground.

Any equitable and practical leasing system must recognize that the discovery of ore involves an expenditure of time and labor so that the lessee must be adequately protected in his contract after a discovery is made. There is rarely any objection by the lessor to have the work continued as long as the lease remains unproductive. Difficulty sometimes arises, however, when the value of the lease is demonstrated. As a suggestion a good way to deal with such a contingency would be to date the lease from the time of the first shipment of ore.

There is considerable variation in the size of the blocks of ground granted to lessees. Old mines worked on the leasing system may be divided into a number of blocks, which are worked by different sets of lessees. The size of the blocks may be 100 feet along the strike of the vein and the height between two levels or the entire block of ground between two successive levels. Sometimes the lessors reserve the right to work blocks of ground immediately above or below any new finds made by the lessees in their explorations.

An entire mine or prospect, including all the surface and underground equipment, may be leased. If the mine or prospect is not equipped with machinery the lessee usually is required to furnish it. A fair medium between this arrangement and one where the lessor furnishes the equipment may be arranged whereby each party furnishes part of the equipment.

BASES OF SETTLEMENT (ROYALTIES)

In the leasing system the royalty represents the division of the profit on the ore produced. It is to the advantage of both parties to the contract to make an equitable adjustment of royalty payments. High royalties put a premium

on gouging for higher grade ores and decreases the lessees incentive to do exploration work.

There never has been any attempt at standardization of royalty payments, as the problem involves not only variation of the metal content of ores but also variable costs of mining, treatment, and transportation. In consequence, royalties must be adjusted to suit the conditions at each property. In general, three methods are employed in settling royalties in mine leasing. These are flat royalty, split-check, and sliding-scale methods of payment.

With the flat royalty method of payment the royalty to be paid is determined largely on the amount of ore available and the difficulties of exploration or development. Royalties may be 5 to 30 percent of the assay value of the ore or of the net returns from the mill or smelter. The higher royalties usually indicate that high-grade ore exists under favorable conditions for mining, while the lower royalties indicate the reverse. Conditions are so variable in different mines that no general rules can be applied in fixing royalties.

As an illustration of the flat scale method of royalty payments the Goldfield district, Nevada, may be cited. This district owed its rapid expansion in its initial stages of development almost entirely to the leasing system, and with the exception of the Combination mine virtually all of the ore bodies were discovered by lessees. The early leases in the district generally ran for 1 year, and the royalties ranged from 20 to 30 percent of the returns from the smelter. So rich were many of the ore bodies that numbers of successful lessees made fortunes in a few weeks. The original Florence lease produced \$750,000; the Bowes and Kernick lease on the Jumbo property, \$500,000; the Reilly lease on the Florence No. 2, \$550,000; and the Hayes and Monette lease, a gross total of \$4,400,000.

The silver-gold deposits of the Tonopah district were also prospected and developed in the early years by leases let on a flat scale basis. To prove the value of the original discovery in this district, Jim Butler, the discoverer, gave a number of leases with a flat royalty of 25 percent based on the smelter returns. These leases ran 1 year, and many were very profitable. Ore valued at \$4,000,000 was extracted by lessees before the leases expired. It is interesting to note that all the leases given by Butler were verbal, and even though the property was sold while a number of them were still in effect, the verbal contracts made were observed to the letter.

With the split-check system of leasing the net returns from the mill or smelter are divided equally between the lessee and the lessor. In this arrangement the lessor furnishes the equipment for mining, explosives, and all other supplies necessary, while the lessee furnishes the labor and supervision. By the split-check system the lessees are able to do more prospecting and development work than is possible with other methods of royalty payment since there is closer cooperation between lessee and lessor.

The split-check system is employed extensively in subleasing where a lessee may take a lease on an entire mine on a flat royalty basis and sublease blocks of ground to smaller operators. The original lessee usually provides all the equipment and supplies, so that the sublessee risks only his time and labor. Many experienced lessees prefer the subleasing arrangement, even though it entails higher royalty, because they prefer to work with machine drills rather than hand steel or other less efficient equipment.

An example of the split check system of royalty payments practiced in the Cripple Creek district is described by Keeler.² The Engineers Lease, Inc., organized by a group of engineers, obtained leases in the Vindicator, Cresson, and Portland mines of the Cripple Creek district, on royalty terms whereby the returns from the mill were divided equally between the lessee and the lessor. Supplies and equipment were furnished by the companies, and the Engineers Lease furnished all the labor, management, and engineering skill. The split-check principle was carried farther in the division of the profits accruing to the Engineers Lease: two fifths of the profit is paid to the miners, who in addition were guaranteed a "grubstake" wage of \$1.25 per day, two fifths is paid to the men furnishing the capital for paying the miners, and one fifth goes to the organizers of the plan.

In the sliding-scale system of royalty payments the royalties are low on the lower-grade ores and become higher with increasing value of the ore. The justification for fixing a ratio between the value of the ore and the royalty is that the lessee can better bear the higher royalty on the higher-grade ore. Were no sliding scale adopted and all ores, irrespective of grade, charged at an average royalty, that royalty would be so high as to make the production of lower-grade ores unprofitable to the lessee. Hence, a low royalty on low-grade ores stimulates mining this class of ore. An illustration of sliding-scale royalty payments is as follows:

<u>Value of ore per ton.</u>	<u>Royalty payment - percent</u>
Up to \$10	10
\$10 to \$20	15
\$20 to \$40	20
\$40 to \$100	25
\$100 to \$240	30
\$240 and over	50

With the sliding-scale system, royalties vary greatly in different mines. The system is particularly applicable where there is considerable variation in the metal content of the ores. As high as 95 percent royalty has been paid by a lessee on exceptionally rich ore, and the lessee was still able to make a profit.

A variation of sliding-scale royalty has been employed whereby the percentage of the gross value of the ore is determined by taking a fractional part of the value of the ore. Thus, if the fraction decided upon is one third and the ore assayed \$60 per ton, the royalty would be one third of 60 or 20

² Keeler, Ralph, Unique Unemployment Relief, Min. Jour., vol. 15, no. 21, 1932, p. 6.

percent, equivalent to \$12 per ton. Obviously, if the ore assayed \$300 per ton the royalty would be one third of 300 or 100 percent, the entire value of the ore. To avoid this difficulty provision is made that the royalty does not exceed a fixed percentage of the higher-grade ore.

UNITED STATES BUREAU OF MINES
SCOTT TURNER, DIRECTOR

INFORMATION CIRCULAR

HOW CAN THE BUREAU OF MINES BEST SERVE MINING?
(THE BUREAU AND THE SOUTH)



BY

MILTON H. FIES

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May, 1934

INFORMATION CIRCULAR

UNITED STATES BUREAU OF MINES

HOW CAN THE BUREAU OF MINES MOST EFFECTIVELY SERVE MINING? ¹
(THE BUREAU AND THE SOUTH)

By Milton H. Fies ²

To present some idea as to how the Bureau of Mines may most effectively serve mining, it is necessary that we make some résumé of its past accomplishments in order to determine what, if any, changes will be of greatest service to the industry and to the nation.

There are a great many times in life when one begins to wonder whether or not Emerson, in his famous epigram about "mousetraps" and "beaten paths" and "final recognition of intrinsic worth", took into account the item of time. The intrinsic value of any movement, of any work, of any accomplishment will ultimately be recognized, of that there is no doubt, but many a man in history has gone to his end in martyrdom because the world was slow in the acknowledgment of the true excellence of his worth, and this is likewise true of institutions.

All of us, I am sure, are unanimous in one criticism of the Bureau of Mines and that is that it has been too modest. If there is any Department of the Government in Washington today that is in need of a publicity agent, it is the Bureau of Mines, and, if such a press agent were established in the Bureau, I know of no one in a similar post in Washington who would have a finer field in which to display his talents.

But, the Bureau of Mines has no publicity agent, and it is my understanding that its function in the Government has been to some extent minimized and that there is a possibility of its being relegated to some subordinate position within some other Bureau. And I am here from the South - that portion of our country which has ever been the protestant thruout our nation's history - to remonstrate, in a spirit of reason and in a spirit of fairness toward the Bureau of Mines, against any such movement.

In his book, "Looking Forward"; the President of the United States makes the statement "that the achievement of good government is a long, slow task, persuading, leading, sacrificing, teaching always, because perhaps the greatest duty of statesmanship is to educate". That word "educate", as I take it, was used by him in a broad sense, and while there probably is

¹ Reprinted from The Mining Congress Journal, January 1934 (vol. 20, no. 1, pp. 40-42).

² Vice president, DeBardeloben Coal Corporation, Birmingham, Ala.

a vast difference between the work of a coal miner, an oil driller, or an ore mucker, and the duty of a statesman, the education of the coal miner and the oil driller and the ore mucker along the right road makes easier the work of the statesman.

No man questions the fact that it is the duty of the government to educate. As far as the South is concerned, there has been no agency of the Federal Government that has contributed as much toward the education of the mass of the workers in industry as has the Bureau of Mines. The volume of knowledge which the Bureau has carried to the citizenship of this country today has by no means been confined to that which interests the worker, nor has the benefit derived been limited to this class of our people. Its schools for superintendents and foremen, its technical research, its economic branch, mineral statistics, experimental stations, and its explosives experimentation, have provided a source of knowledge for the management of interested industries which is without parallel in the history of any other similar Bureau in the world.

From the view-point of the South, the Bureau of Mines in the future can most effectively serve in that section by a continuation of the sound policies which it has pursued in the past. I do not mean to convey the impression that it cannot improve in some instances the character of its services, as I shall point out later, but the point I wish to emphasize is that fundamentally and for results achieved, the work of the Bureau during its past history has been effective and vital.

Let us consider the matter of the Bureau of Mines from a less-provincial standpoint and regard its achievement in the important matter of national safety in industry. In the Safety Division alone of the U. S. Bureau of Mines, more than 755,000 (or over three-quarters of a million) men have been trained in first-aid and mine rescue work, 400,000 of them having been trained in the last four years.

To June 30th, of this year, 100 percent first-aid certificates had been issued to 1,037 industrial plants. In the year ending June 30, 1933, 230 such certificates involved the training of 38,852 persons. During that same year 958 mining officials completed the accident-prevention course, and nearly 600 others took portions of the course. Since the inception of this course, 4,411 bituminous coal-mine officials have completed the full course.

During the fiscal year ending 1933, first-aid instructors' certificates were issued to 625 men in Pennsylvania, 161 in Kentucky, and 103 in Alabama. First-aid training given to from 70,000 to 100,000 men annually in industry has resulted in the saving of at least 200 lives annually, and many reliable industrial organizations place the ratio of the relative safety of a first-aid-trained man to an untrained man at from one to four to as much as from one to eight.

If I may incidentally refer to another section of the country, you have no doubt noted the splendid progress shown by the State of Pennsylvania. And, without any thought to in any way disparage the fine work of the Department of Mines for that State, I think it may safely be stated that through the cooperation of Mr. Glasgow and the Bureau of Mines the added training of workers in the State of Pennsylvania has aided in making possible the wonderful accomplishment in that State in the past few years.

Consider the accident record of the United States as compared to the larger coal-producing foreign countries, such as Great Britain, France, Belgium, and Prussia. Our country shows more favorably on basis of fatalities per million short tons of coal produced, but our record on basis of severity, that is, per thousand 300-day employees, is not as satisfactory as in the countries mentioned.

The unfavorable comparison on basis of severity and the favorable comparison on basis of tons produced can be explained, I think, to some extent, on the ground that we produce more coal per man, have larger seams, greater amount of machinery, etc., which condition helps our fatality record on basis of tons produced and at the same time harms our record on basis of severity, for the reason that we have fewer men employed to produce a given number of tons. We should not be satisfied with the situation as it relates to our severity record; and its improvement, with all its humane aspects, can be realized largely thru the activities of a Federal agency that serves the whole country, and the Bureau of Mines is the agency to bring this about.

It is not my purpose to narrow this discussion to mining. The Bureau of Mines might well be termed a "Bureau of Industry", because it has not confined its efforts to mining alone, but, throughout the nation, wherever and whenever called upon, it has given of itself freely, for all types of industry, whether it be refineries, cement plants, steel mills, or other types of manufacturing plants, particularly as to safety.

But, to get back to the South. Consider the results obtained along safety lines in the States of Alabama, Florida, Tennessee, Louisiana, Georgia, Mississippi, and North and South Carolina, wherein the activities of the Bureau have been marked by splendid achievement.

In Georgia, Alabama, and Tennessee, alone, for the year 1922, the estimated cost of accidents in coal mines was \$1,060,000, or $4\frac{1}{2}$ cents per ton of coal produced. In 1932, the cost was \$135,000, or, roughly, 1.2 cents per ton. From July 1, 1910 to June 30, 1933, in Alabama there were 43,351 trained in first aid and rescue, and in that same period in the three States mentioned, including Alabama, there were 70,988 trained in first aid, 4,501 in mine rescue, or a grand total of, roughly, 75,000. In these same States, from 1930 to 1933, inclusive, there were 130 plants that made it compulsory for 100 percent of their employees to be trained in first aid. From the year 1931, to date, the Bureau of Mines has trained in these Southern States, as instructors - men qualified to help others - a total of 501.

The average number of fatalities in Alabama coal mines from 1922 to 1926, inclusive, was 129, with 170,000 tons produced per fatality; 1927 to 1931, inclusive, there were 64 fatalities per year, with average production of coal per fatality at approximately 291,000 tons; whereas, in the year 1932, there were only 18 fatalities with production of 472,000 tons per fatality.

Assuming the cost of a fatality to be \$5,000, first-aid training effects a saving of \$1,000,000 annually to mining companies. In Alabama, for the period 1922-1926, on this basis the annual average cost of coal-mine fatalities was \$645,000; for the period 1927-1931, the cost was \$320,000; and in 1932, \$90,000. The efforts of the Bureau in cooperation with the State Mining Department and the Alabama Mining Institute were largely responsible for this annual saving of \$325,000 in the first two periods mentioned.

Recent Bureau of Mines statistics indicate that one person in every six engaged in mining is injured every year. Therefore, about 16,500 of the 100,000 persons trained in first aid each year should receive an injury. If training reduces accidents 35 percent, which we would judge to be a conservative estimate from the ratios given above, the annual reduction would be 5,600, due to the Bureau's first-aid training. Assuming a cost of \$100 per accident, the saving is about \$560,000. In Alabama, in 1932, it may have been expected that 3,500 persons would be injured and the average cost would be \$350,000. If 35 percent of the 5,836 persons trained in first aid were prevented from receiving an injury, then the reduction for 1932 would be 2,042 less injuries and at \$100 per accident the estimated saving is \$204,200. First-aid training was then responsible for this saving in Alabama alone.

The savings in money and misery accrue, not only to mine employees but to the mining industry and to the general public, through first aid rendered by Bureau-trained men.

The economic saving and the accidents prevented through the instruction in accident prevention, through the safety inspections and reports, through the special investigations, and through the analysis of mine air, dust, and coal samples, cannot be estimated; but the cooperation of the Bureau of Mines with the States, mining societies and institutes, and the individual companies has aided materially in reducing cost of accidents and mining. The value of organized safety, such as advocated by the Holmes Safety Association, has been widely acknowledged, though the cost has not been evaluated to show the saving to the industry.

The work of the Holmes Safety Association in the South has been of special interest, particularly in the rural sections at isolated mines and it has, moreover, been of unusual value to the negro miner. It will be interesting to you who come from other sections of the country to know that the negro Holmes chapters, held separately of course from the white chapters, are looked upon by the negro in somewhat the same manner as he looks upon his lodge. They take the work seriously and it has been of great educational worth to them.

I do not mean to depreciate the contribution of the Holmes Safety Association to our native white miners in the South, but it has been an interesting study, as far as race psychology goes, to us who are interested in the negro, to observe his attitude.

There is one company attentive to its safety record, that has made a comparative study of accidents to negroes and whites. At its mines, at which about 35 percent of the total number of employees are negroes with about 50 percent of the face workers negroes, for the years 1931, 1932, and through November 1933, the following record is interesting:

	Percent	
	White	Colored
Employees	65	35
Disabling injuries	73	27
Compensable injuries	76	24
Noncompensable injuries	69	31
Days lost	81	19
Cost of compensation	83	17

The safety bulletins issued by the Bureau have been instructive not only from a safety standpoint but, rather, informative generally. Many of our underprivileged men have been able to learn about mine gases, mine timbering, and other valuable technical data which they would not have been able to acquire without the assistance of the Bureau.

It may not be amiss to state that the Holmes Safety Association work was inaugurated in Alabama in 1923 and at that time there were approximately 40 chapters organized, which, if I recall correctly, constituted a majority of all such chapters in all the States of the Union. On June 30, 1933, there were 58 chapters operating in the State.

I trust that I may be pardoned for the pride I possess in Alabama's safety achievement, and while I do not propose to bore you unduly with statistics, I think that the table showing the comparison of Alabama fatality rates with those of the United States as a whole, from 1911 to 1932, inclusive, is a matter of much interest as it relates to the Bureau of Mines, for the reason that since its inception I doubt if there is a State in the Union that has appreciated its worth to any greater extent than has the State of Alabama. Every individual therein interested in coal mining knows of the great contribution made by the Bureau in the reduction of fatalities due to gas and dust explosions in coal mines. Its research work and its method of education among the mining men of the State have been invaluable.

Comparison of Alabama fatality rates with
that of the United States as a whole,
1911 to 1932, inclusive

Year	Number killed		Deaths per million tons		Tons per fatality	
	Ala.	U. S.	Ala.	U. S.	Ala.	U. S.
1911	209	2,656	13.92	5.35	71,827	186,887
1912	121	2,419	7.35	4.53	136,471	220,945
1913	124	2,785	6.92	4.89	144,413	204,685
1914	137	2,454	8.82	4.78	123,222	209,261
1915	63	2,269	4.13	4.27	242,331	234,297
1916	118	2,226	6.47	3.77	154,531	265,094
1917	109	2,696	5.34	4.14	187,298	241,618
1918	110	2,580	5.63	3.80	177,473	262,873
1919	93	2,323	5.83	4.19	171,277	238,464
1920	78	2,272	4.48	3.45	222,967	289,729
1921	80	1,995	6.14	3.94	162,668	253,832
1922	183	1,984	9.76	4.16	102,501	240,399
1923	89	2,462	4.25	3.74	235,014	267,223
1924	77	2,402	3.92	4.20	254,752	237,974
1925	162	2,234	7.93	3.84	125,979	260,461
1926	139	2,518	6.46	3.83	254,739	261,241
1927	93	2,231	4.66	3.73	217,106	267,978
1928	67	2,176	3.71	3.78	269,494	264,749
1929	72	2,187	3.91	3.59	255,667	278,380
1930	61	2,014	3.81	3.79	262,416	263,869
1931	26	1,430	2.15	3.27	470,000	296,000
1932	18	923	2.17	3.02	460,000	331,167
<hr/>						
Total	2,229	49,236	127.74	38.06	4,602,146	5,577,126
<hr/>						
Average	101	2,238	5.81	4.00	209,188	253,506

Consider the Bureau's accomplishment in the reduction of fatalities from dust and gas explosions, from the standpoint of Alabama again. In the ten years, 1911-1920, there was an average of 34.3 fatalities per year from this cause; for the period 1921 to 1930, inclusive, 26 fatalities per year, and for the last three years there have been only 3 fatalities from this source, all in the year 1931.

Consider this phase of the Bureau's accomplishment from the standpoint of Alabama, again. In the ten years, 1911-1920, there was an average of 34.3 fatalities per year from gas and dust explosions; for the period 1921 to 1930, inclusive, 26 fatalities per year; and for the last three years there have been only 3 fatalities from this source, all in the year 1931.

Not only has the Bureau, in its study of gas and dust explosions, rendered effective aid to all interested in coal mining in America, but its efforts have contributed much toward the reduction of accidents from this source among all the nations of the earth where coal is mined.

Here is an organization which has served the mining industry, petroleum and allied interests, and has rendered aid to all industry when called upon, at a minimum expense to the Government. The value of mine and metal products is approximately six billion dollars annually. The expense of the Bureau's operation as compared to the annual production value of the industries it serves is, I believe, less than any other department of the Government.

Surely, those who have in mind reducing the services of the Bureau of Mines, either by reduced expenditure within the Bureau or curtailing its personnel, do not realize that never in its history has the Bureau a greater opportunity to render valuable service to industry and the nation along economic lines than at the present time. The whole setup of the successful working out of the Coal Code depends largely upon the establishment of differentials in the various grades of coal based upon the quality of the many seams. These quality values must be determined by sampling and analyses of the various seams, and for this service the agency in which all coal operators have the greatest confidence is the Bureau of Mines.

We are living in an age of so-called long-range planning. The disruption of the relation of production with consumption and mal-distribution of products is the basis of our ills. The statistical department of the Bureau, with its experience and its compiled data as they relate to all mining, is essentially invaluable. As far as coal is concerned, there is no agency anywhere in the land that has even comparable knowledge as to market conditions, competitive fuels, distribution of fuels, and all those valuable facts which can aid in making possible our national recovery as it relates to these great basic industries and to the part which they will play in the Nation's recovery.

There can be no long-range planning for the coal industry without recognition of the fact that in the chemistry of coal lies the ultimate salvation of the industry. Unless our Government is alive to the possibilities in the relationship of coal to organic chemistry, the future progress of this country may be seriously impeded. The research department of the Bureau of Mines has not only kept pace with the investigations conducted in foreign countries, where more extensive research has been carried on, but has contributed in no small measure to research work in this country.

But, since I am to suggest the most effective manner in which the Bureau can serve mining, I would stress this point more than any other. There should be an enlargement in its activities and its policy as it relates to research in the chemistry of coal.

There is very little to criticise in the scope and general plan of the Bureau of Mines. Certainly all of us agree that fundamentally its conception of its own duties is sound. And, as stated, it needs, in my opinion, only stressing and emphasizing and enlarging the work which it has done, rather than any radical change in its policies.

Summarizing, briefly, intensive work in research, continuation of its fine safety and health policies and the enlarging and broadening of its statistical bureau, including comparative analyses of various coals in the several fields, will be of immeasurable value to industry and through it to the nation.

It may not be amiss to state, and I think that as an engineer I may be able to make the statement without criticism, that engineers in a great many instances are faddists and, frankly, present quite a few abnormalities. In other words, many of us - to be charitable - possess certain eccentric characteristics, and I think this describes adequately engineers in and out of the Bureau, particularly out of the Bureau. And, where there has been professional difference of opinion in the past, due to such individual eccentricities, we should guard against them in the future so that we may obtain the full measure of value from the Bureau. We have been too prone to criticise unimportant details, prompted largely by personal feelings, and we should be big enough to cooperate with the Bureau on the basis of its broad ideals and forget our own, and their, idiosyncrasies.

Those of us in the South who believe in the Bureau, and we are legion, stand ready now and at any time to carry on the struggle for its individual existence with all the fullness of our strength.

UNITED STATES BUREAU OF MINES
SCOTT TURNER, DIRECTOR

INFORMATION CIRCULAR

MINING LAWS OF BRITISH GUIANA



BY

E. P. YOUNGMAN

INFORMATION CIRCULAR

UNITED STATES BUREAU OF MINES

MINING LAWS OF BRITISH GUIANA 1

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1 The Bureau of Mines will welcome the reprinting of this paper, provided the following footnote acknowledgment is used: "Reprinted from U. S. Bureau of Mines Information Circular 6777."

2 Formerly of Rare metals and nonmetals division, U. S. Bureau of Mines.

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PART I. - GENERAL REGULATIONS

FOREWORD

This paper presents one of a series of digests of foreign mining legislation and court decisions that is being prepared in advance of a general report relative to the rights of American citizens to explore for minerals and to own and operate mines in various foreign countries. This interpretation of the mining laws of British Guiana has been prepared from the best available sources in Washington and is released subject to amplification and correction, if necessary, by United States foreign-service officials.

INTRODUCTION

The sources of information available in the preparation of this paper are the following:

1. The Crown lands ordinance, No. 32 of 1903, September 26, 1903.
2. The lands and mines department ordinance, No. 31 of 1903, September 26, 1903.
3. The mining (consolidation) ordinance, No. 34 of 1920, March 31, 1921.
4. The mining ordinance, 1931, amending the mining (consolidation) ordinance, October 6, 1931.
5. The mining regulations, 1931, August 5, 1931.
6. The bauxite mining regulations, 1930, October 1, 1930.
7. The mining (oil and natural gas) regulations, 1931 (draft published January 31, 1931, but not so far adopted).
8. The companies (consolidation) ordinance, No. 17 of 1913, August 30, 1913.

Coal does not seem to fall under any of the foregoing regulatory ordinances, although it is included in the consolidated mining ordinance. The general mining regulations of 1931 specifically exclude coal, as do also the oil and natural gas draft regulations.

RIGHTS OF FOREIGNERS

Foreigners are actively interested in mining operations in British Guiana, but the exact legal status of foreign rights is not clear. However, as far as actual enactments go, it would seem that, except with respect to petroleum and bauxite, no discrimination is made against foreigners or foreign capital.

Several attempts have been made to keep out of British Guiana non-British capital. ³, ⁴, ⁵ The aliens (regulation of land holding) ordinance (proposed in 1921), for example, would have prohibited an alien or an alien company from holding land of more than 5 acres without the permission of the Governor, and foreign holdings would have been forfeitable should an individual or a company offend against the ordinance. Local opposition apparently killed this bill. At least as early as the summer of 1929, local displeasure over the embargo upon foreign capital in the exploitation of oil in British Guiana manifested itself in public meetings and resolutions by influential citizens, merchants, chambers of commerce, and the elected members of the Legislative Council. ⁶, ⁷ A protesting resolution of the elected section of the Legislative Council, March 14, 1930, reads:

That the local government be requested to introduce forthwith the necessary ordinances and regulations whereby the natural resources of the colony, be they mineral or otherwise, will be open to location and exploitation by any persons whatsoever and of whatever nationality, subject only to such colonial or other rights in times of national emergency as may be deemed necessary.

A new draft of the mining (oil and natural gas) regulations, 1931, published in the Official Gazette of January 31, 1931, but as far as the most recent reports show not yet ratified as law, reflect this local agitation. The new draft omits entirely the clause of the 1912 regulations prohibiting the issuance of licenses and leases to any other than a British company;

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- ³ Davis, Chester W., Newspaper Comment on the Aliens (Regulation of Land Holding) Ordinance Proposed in British Guiana: Consular Rept. 45537, Georgetown, Dec. 20, 1921, Bureau of Mines foreign file 3078.
 - ⁴ Davis, Chester W., Aliens (Regulation of Land Holding) Ordinance in British Guiana: Consular Rept. 47088, Georgetown, Jan. 16, 1922, Bureau of Mines foreign file 3078.
 - ⁵ Davis, Chester W., Progress of the Aliens (Regulation of Land Holding) Ordinance in British Guiana: Consular Rept. 45703, Georgetown, Jan. 2, 1921, Bureau of Mines foreign file 3078.
 - ⁶ Brown, Harold R., Embargo on Foreign Capital Employed in the Exploitation of Oil in British Guiana: Consular Rept. 314704, Georgetown, July 29, 1929.
 - ⁷ Brown, Harold R., Serious Economic Flight of British Guiana, Consular Rept. 333594, Georgetown, March 15, 1930, Bureau of Mines foreign file 10775.

and it provides (in section 18 of the mining lease) that in the event of war or other national emergency the Governor may take control of the works, plant, and premises, and that the lessee shall conform to and obey all the directions issued by the Governor.

The new regulations governing bauxite (law of October 1, 1930) make no ruling with respect to aliens except that a prospecting permit may not be issued to any resident outside the colony unless such person is represented by a duly constituted attorney resident in the colony and unless a power of attorney is deposited in the deeds registry. The prescribed mining lease contains practically the same provision with respect to the Governor's right to preempt bauxite in times of national emergency as that mentioned in the preceding paragraph with respect to oil. The Governor is given the further right to prohibit absolutely the removal of any bauxite products from the colony. No lessee may, without the written permission of the Governor, supply to any alien any bauxite or product thereof, whether within or without the colony.

OWNERSHIP

The consolidated mining law of 1920 established the right of the Government (His Majesty) to any metal or mineral whatsoever in all Crown lands granted subsequent to the passage of the mining ordinance of 1903.

Notwithstanding anything in this or any other ordinance contained, no grant of Crown land made after the passing of the mining ordinance, 1903, shall be deemed to confer upon the grantee any right to any mineral or metal whatsoever therein, and all those minerals and metals shall remain the absolute property of His Majesty. (Art. 53-6, No. 34 of 1920).

Other sections of the 1920 ordinance (arts. 2, 13, 14, and 15) make it plain that "all minerals and metals" include gold, silver, valuable minerals, precious stones, mineral oil, asphalt, coal, and like substances; and that, moreover, "minerals and metals" include their ores and compounds.

Although the Crown lands ordinance, No. 32 of 1903, made the same stipulation with respect to the State's ownership of "all minerals," the term "mineral" had a different interpretation. The exact rights of the Government to different minerals upon lands under prior statutes are discussed in a report by the Commissioner of Lands and Mines of British Guiana in 1926.⁸ This report was occasioned by the increasing need for a ruling upon the question as to whether petroleum is a mineral and by the fact that no statute covered the rights of the Government to issue licenses to search for mineral oil on privately owned lands granted prior to 1920.

⁸ Blake, Gilson G., jr., Laws of British Guiana Concerning the Right to Minerals and Mineral Oil in Privately Owned Lands: Consular Rept. 216761, Georgetown, June 29, 1926, 8 pp.

The Imperial Institute's volume upon the mining laws of British Guiana,⁹ also deals with the question and history of the Government's ownership of minerals more fully than is possible here.

With respect to the lands of the colony, the consolidated law of 1920, as amended by the ordinance to amend the mining consolidated ordinance, No. 20 of 1931, in article 52 (2), provides that --

The Commissioner may, with the approval of the Governor, issue a license to any person authorizing him to enter on lands of the colony and there search and mine for, take, and appropriate gold, silver, and valuable minerals, precious stones, and other metals and minerals, mineral oil, asphalt, coal, or other substance of a like nature.

With respect to private land, article 52 (1) of the same law makes the same provision, except that the right to grant licenses for mineral oil, asphalt, coal, or like substances is given only for private lands granted since the mining (mineral oil) ordinance of 1910. Article 53 (1) of the consolidated ordinance of 1920 gives the owner of lands granted prior to the passage of the mining ordinance of 1903 the right to enjoy all metals other than gold and silver,¹⁰ together with the right to appropriate them to his own use without obtaining a license therefor.

However, nothing in the section just quoted shall give the owner of private lands any right to any precious stones therein or thereon. (Art. 53-5, No. 34 of 1920). Also, nothing in this section shall authorize the owner to appropriate any "valuable mineral" on his land without obtaining a license. (Art. 53-4, No. 34 of 1920).

Other more or less relevant provisions are the following:

The Department of Lands and Mines has charge of and is guardian over all Crown lands, forests, rivers, and creeks of the colony, except forest reserves. It has charge also of all lands that are the property of the colony, except lands occupied by buildings under charge of the Public Works Department. (Arts. 5 and 8, No. 31 of 1903).

The Commissioner has charge of and is guardian over all metals and minerals, including gold, silver, and valuable minerals, precious stones, mineral oil, asphalt, coal, and other substances of a like nature within the lands, rivers, and creeks of the colony. (Art. 3, No. 34 of 1920).

Nothing in any concession or lease shall be construed to prevent the Governor from directing at any time that any portion of the land with respect to which it is granted shall be taken and used for public purposes; and when the

⁹ Imperial Institute (British), The Mining Laws of British Empire and Foreign Countries; British Guiana: Vol. 9, London, 1927, pp. 3-5.

¹⁰ Slight modification of this exception is made in sections 2, 3, and 4, art. 53, No. 34 of 1920.

Governor so directs, the land specified in the order of the Governor shall be taken and used for public purposes without giving any right to compensation therefor. (Art. 23, No. 34 of 1920).

Whenever a concession or lease is granted with respect to unoccupied Crown lands, the Governor may make an absolute grant of the Crown lands with respect to which the concession or lease was granted without requiring compliance with the Crown lands ordinance and the Crown lands regulations in force at the time, provided that no grant shall be made under this section without the approval of the Secretary of State. (Art. 18, No. 34 of 1920).

ADMINISTRATION

The Lands and Mines Department Ordinance (No. 31 of 1903), creating the Department of Lands and Mines, provided for a Commissioner of Lands and Mines and for such assistants, surveyors, clerks, and other officers as the Governor might from time to time appoint. The Commissioner and every other officer is subject not only to the duties prescribed by the law but also to any duties required by the Governor. (Arts. 1, 2, and 3, No. 31 of 1903).

The Governor, in Council, may make and when made may alter, amend, or revoke regulations with respect to any or all of the following matters (Art. 98, No. 34 of 1920):

1. The prospecting and locating of claims and the issue of concessions, leases, and licenses.
2. Filing of objections.
3. Duration of concessions, leases, or licenses, and size and extent of claims.
4. Mode of working claims and deposit of waste and tailings.
5. Keeping of records of gold, silver, and valuable minerals, precious stones, metals, minerals, mineral oil, asphalt, coal, or other substances found.
6. Definition of boundaries and maintenance of boundary marks.
7. The amalgamation, purchase, and transfer of concessions, leases, and licenses.
8. Surveys.
9. The regulation of the use of the surface of the land, and of the cutting of timber and fuel.
10. Water rights and the use of water.

11. Permits to occupy land for business purposes connected with mining.
12. Cutting of paths and trails, and the use of them.
13. Jumping of claims.
14. Employment and registration of labourers, the payment of wages and the duties of employers.
15. The conveyance, detention, sale, and purchase, of gold, silver, valuable minerals, precious stones, metals, minerals, mineral oil, asphalt, coal, or other substances of a like nature.
16. Duties of traders in gold, valuable minerals, or precious stones.
17. The sanitary regulation of claims and places adjacent thereto.
18. The regulation of mines.
19. The determination of disputes.
20. Employment and rights of aboriginal Indians.
21. Disposal of forfeited gold, silver, or valuable minerals, or precious stones, or metals, minerals, mineral oil, asphalt, coal, or other substances of a like nature.
22. Prescribing the conditions under which mining rights in respect of the same area may be granted to different persons and providing, where necessary, for the payment of compensation to the first occupier for disturbance of his rights.
23. The determination of royalties due, fees of office, and other amounts payable to the Colonial Government.
24. The regulation of the conditions on which entry may be made on private lands and securing to those entitled thereto due compensation for disturbance of the surface of the land.
25. The grant to first discoverers of new payable mining fields of award claims or any payment out of funds provided by the Legislative Council for the purpose.
26. All matters not hereinbefore specially mentioned connected with the search or mining for or dealing with gold, silver, valuable minerals, or precious stones, or metals, minerals, mineral oil, asphalt, coal, and other substances of a like nature, and generally all matters connected with the proper carrying out of the provisions of this Ordinance.

The Governor may, by notice in the Gazette and in a newspaper of the colony, declare any portions of the colony to be mining districts. He may establish one or more stations in or near a mining district and direct that all persons going to or from that district report at one or more specified stations for examination and search. (Art. 4, No. 34 of 1920).

The Governor may appoint for each mining district a warden and also sub-wardens or other officers, including female searchers. In addition to being responsible for the efficient working of his district, a warden may be required by the Governor to exercise the duties of a magistrate (which duties shall not conflict with those of the magistrate of the judicial district of which the mining district forms a part. (Arts. 5, 6, 7, 8, 9, and 10, No. 34 of 1920). The Commissioner may perform any of the powers conferred upon a warden or any other officer except the magisterial duties just mentioned. (Art. 11, No. 34 of 1920). A warden may at any time inspect any claim or building or work connected therewith. (Art. 12, No. 34 of 1920).

REGISTRATION

Of registration, the bulletin of the Imperial Institute 11 says:

British Guiana, as is the case with other dominions and colonies subject to Roman-Dutch law, has a highly developed system of deeds registration, the statutory provisions in relation to which are to be found in the deeds registries ordinance, 1919, as amended by the deeds registry ordinance, 1919, and amendment ordinance, 1925.

It is not proposed to consider the wide subject of deeds registration in this work, except to observe that certain leases or deeds or assignments of the rights thereby created or charged must, in order to be valid as against a subsequent transferee or chargee for value, be registered in the deeds register.

With regard to the registration of mining licenses, permits, or grants, special reference should be made to the Department of Lands and Mines Ordinance, 1903, section 7, which requires the Department of Lands and Mines to keep proper registers of all grants of Crown lands or any licenses connected therewith, while section 13 confers the right to inspect the registers during office hours on payment of a small fee. Any person that satisfies the Commissioner of Lands and Mines that he has an interest in the same may obtain a certified copy of any register or document in the possession of the Department of Lands and Mines on the payment of a small fee.

Further, the Department of Lands and Mines keeps a proper and separate register of all lands that are the property of the colony.

11 Imperial Institute (British), Work cited, p. 41.

Under the mining regulations, the question of registration of title is not considered. The following matters must be registered or filed:

1. Applications for the issue of a prospecting license. (Art. 5-1, reg., 1931).
2. Reports of issues of prospecting licenses. (Art. 5-2, reg., 1931).
3. The address of the applicant for a prospecting license. (Arts. 3-2 and 5-3, reg., 1931).
4. Notices of location of claim and application for claim license. (Art. 14 and 15, reg., 1931).
5. Application for a concession or lease. (Arts. 43-5, reg., 1931).
6. Transfer of a concession or license and address of transferee. (Art. 51, reg., 1931).
7. The fact of a purchase or a sale at execution of a holder's rights under a concession, lease, or license. (Art. 52, reg., 1931).
8. Application for an exclusive bauxite permission. (Art. 3-2, bauxite reg., 1930).

PART II. - MINERALS OTHER THAN BAUXITE, MINERAL OIL, AND NATURAL GAS

GENERAL

Except upon private land obtained before the passage of the mining ordinance of 1903, no mineral rights may be exercised without the sanction of the State. (Art. 53, No. 34 of 1920). (See preceding section of this paper entitled "Ownership.")

A prospecting right does not necessarily precede a mining right, as the Governor, with certain restrictions, may grant to "anyone" a mining right, as well as an exclusive exploration right. (Art. 13-15, No. 34 of 1920).

EXPLORATION RIGHTS

Exclusive Permission to Explore

General.- The Governor in Council may issue an exclusive permission to anyone to occupy temporarily any unoccupied Crown land for the purpose of testing the value thereof but not for profit. (Art. 15, No. 34 of 1920). However, during the continuance of an exclusive permission to explore, the holder may mark off such areas as he may desire to work for profit under a concession or a lease, or he may locate claims within the area. (Art. 35-3-f, reg., 1931). The permittee shall not give out any portion of the land to

lessees ¹² or to tributors ¹³ The terms and conditions set forth in the following paragraphs may be varied by the Governor in Council in any individual case. (Art. 15, No. 34 of 1920).

Application.- The application, addressed to the Commissioner, shall include the nationality of the person for whom the permit is sought, in addition to the proper description of the size, nature and location of the area and the term for which the permit is desired. The filing fee is \$10. (Art. 37, reg., 1931).

Duration.- A permit shall not be granted for a longer period than 3 years; renewals or extensions shall not be for more than 1 year at a time. (Art. 35-3-a, reg., 1931).

Area.- The area shall not be less than 500 acres, to be defined by natural features or as prescribed by the regulations with respect to the marking of boundaries (arts. 7, 8, 9, reg., 1931). All mining claims previously located within the area shall be excluded from the permit and the extent of such claims from the acreage for which rent is payable. The holder of an exclusive permit may from time to time abandon any part or parts of the area, provided that he demarcates such part or parts on the ground and deposits the estimated cost of inspection. When the area to be abandoned has been inspected by the inspecting officer, no further rent shall be payable therefor, and any difference between the amount deposited and the actual cost of inspection shall be refunded. (Art. 35-3b, d, e, reg., 1931).

PROSPECTING RIGHTS

Prospecting Licenses for the Location of Claims

General.- A prospecting license entitles the holder personally or through someone authorized by him, with the written approval of the Commissioner or warden to prospect and to locate claims in every mining district (Art. 4-5, reg., 1931) on any Crown lands of the colony, provided such land has not been previously occupied or lawfully located or reserved by public notice or is not a landing place or a creek set aside by the warden for drinking purposes or building sites or situated within 300 yards of the land boundary of the colony (Art. 6-1, reg., 1931).

Application.- A written application for a prospecting license for the purpose of locating claims, which shall be made to the Commissioner or warden, shall contain (arts. 3-1 and 2, reg., 1931):

1. The full name and address of each person for whom the license is desired.
2. The name, style, or firm (if any) by which the prospecting and working are to be carried on.

¹², ¹³ For definition, see section of this paper entitled "Lessees and Tributors", p. 21.

3. A permanent residential address in the colony, called a "registered address", at which all notices and other processes necessary for the purposes of the regulations may be served.
4. A statement that the applicant is not under 21 years of age.

Qualifications.- A minor is not qualified to receive a license. Any person convicted of an offense against the employees and servants ordinance or any amending ordinance is not entitled to a license. (Arts. 3-5 and 3-6, reg., 1931). It shall not be lawful for any person (except a tributor) while he is employed on or in respect to a claim to apply for, receive, or hold a prospecting license except for his employer. (Art. 3-7, reg., 1931).

The names and addresses of the persons constituting a firm, syndicate, or partnership must be reported to the Commissioner or the warden before a license may be issued. If a license is applied for in the names of more than one person, the license shall be granted to one, on behalf of the others. (Arts. 3-3 and 4, reg. 1931). And only the person authorized under the license or someone empowered by him, in writing, shall be entitled to enter a mining district to prospect or locate claims. (Art. 6-2, reg., 1931).

Duration.- A license shall be in force for 12 months from the date of its issuance. (Art. 4-3, reg., 1931). In case the licensee locates a claim, the prospecting license entitles him to work the ground located from the date of location until the application for a claim license can be published and the claim either granted or refused. (Art. 4-6, reg., 1931).

Reward claims.- A reward claim may be granted by the Governor in Council to the holder of a prospecting license locating, in accordance with the mining regulations, one or more claims (but no more than six to one person in one locality), not less than 10 miles from an existing working claim, and proving to the satisfaction of the Commissioner or warden that the claim contains gold, silver, precious stones, or valuable minerals in commercial quantities.

The holder of a reward claim shall be free from any fees payable under the mining regulations for such a period as the Governor in Council may prescribe; but he shall not be exempt from the payment of royalty.

A reward claim shall remain in existence as long as it is worked to the satisfaction of the Commissioner or warden. Such a claim may be sublet or transferred, but when transferred it shall cease to be known as a "reward claim" and shall be liable to annual license fees. (Art. 28, reg., 1931).

MINING RIGHTS

Mining rights are acquired through (1) a claim license, (2) a mining concession or lease or a mining license, (3) a mining license for private land, or (4) a dredging concession.

1. Claim License

General.- A claim license confers the right of its holder to the use and enjoyment of the surface included within the boundary lines of the claim and of all veins, lodes, ledges, and deposits below such surface and of all the metals, minerals, or precious stones covered by the license, within the vertical planes in which the surface boundaries lie, provided always that the occupation of the land shall be subject to the right of any person to pass through or along any road or path on the land; to the right of any duly authorized person to cut timber or take forest produce on the land; and to the right of any grantee, lessee, or concessionaire, with the approval of the Commissioner in writing, to convey across the land machinery, equipment, materials, or stores required for mining purposes, to construct any road, railway, tramway, or cableway, and to erect and maintain any pipe line, telegraph, telephone, or transmission line or other contrivance in such manner and on such conditions as the Commissioner may approve. (Art. 23, reg., 1931). All gold, precious stones, valuable minerals, or other minerals or metals obtained after location of a claim but before the issuance of the license shall be subject to the same regulations as if they had been obtained after the issuance of the license. (Art. 19, reg., 1931).

Application.- Any one locating a claim shall within a reasonable time (not to exceed 3 months) after the location file at the office of the warden or at the office of the Department of Lands and Mines in Georgetown a notice (in duplicate) in the prescribed form, stating the name of the person for whom the location is made, the name of the actual locator, the names of witnesses in whose presence the location was made, the date of the location, and a sufficiently identifying description of the ground. He shall file an application for a license to mine, in which is stated the minerals to be won. He shall pay the fee prescribed for the filing of the notice and the application, together with the fee for the claim license; otherwise the claim shall be null and void and open to others. (Art. 14-1, reg., 1931). Should a dispute arise as to who is the first applicant, the person that first lawfully located the claim shall be considered the first applicant. (Art. 16, reg., 1931).

Qualifications.- No license shall be issued to a minor. Should the right to obtain a license devolve by will or intestacy to a minor, a license may be issued to his guardian. (Art. 22-3, reg., 1931).

The Governor may direct that a license shall not be issued to any person that has been convicted of an indictable offense or of the larceny of gold or of any offense under the consolidated mining ordinance or the mining regulations, or against whom any unsatisfied judgment for laborers' wages exists, or against whom any judgment or sentence has been given for ill treatment or neglect of any laborer. (Art. 22-4, reg., 1931).

No license shall be granted to a firm, syndicate, or partnership unless the names and addresses of the persons constituting the association are stated in writing, signed by such persons, to the Commissioner. (Art. 22-5, reg., 1931).

Size and shape of area.- Any claim for the mining of gold or precious stones or valuable minerals, or for minerals or metals, shall not be more than 1,500 feet long or 800 feet wide. (Art. 12, reg., 1931).

The side lines and the end lines of a claim shall, as far as possible, run in parallel lines. Where prior locations or natural features prevent, the shape shall be that approved by the warden with the consent of the Commissioner, or by the Commissioner, or by a surveyor making a survey under the mining regulations of 1931. (Art. 13, reg., 1931).

Duration.- A claim license shall remain in force as long as the rent is regularly paid. (Art. 25, reg., 1931).

2. Mining Concessions, Leases, and Licenses

General.- The Governor may, with the approval of the Secretary of State, grant a general concession to anyone entitling him to the soil and the metals and minerals therein. (Art. 13, No. 34 of 1920).

The Governor in Council, on the terms and conditions he considers meet, may grant a mining concession or lease; and the Commissioner, with the approval of the Governor, may issue a mining license, authorizing anyone named therein to occupy any portion of the unoccupied Crown lands and, subject to the provisions of the mining ordinance and the regulations, to mine for and appropriate (arts. 14 and 31, No. 34 of 1920):

1. Gold,¹⁴ silver, and valuable minerals. ¹⁵
2. Precious stones.¹⁶
3. Gold, silver, and valuable minerals, and also precious stones.
4. Any one or more metal or metals ¹⁷ or mineral or minerals ¹⁸ other than gold, silver, valuable minerals, or precious stones.
5. Mineral oil, asphalt, coal, or other substance of a like nature.

¹⁴ "Gold" means raw gold and includes any substance or thing containing gold or of which gold forms a part, whether smelted or not, upon which the payment of royalty is not admitted by the Commissioner, and shall include gold won by milling or dredging. (Art. 2 of No. 34 of 1920 and of reg. of 1931).

¹⁵ "Valuable minerals" means any mineral containing gold, silver, platinum, or any of the rarer metals, or any combination of them together with some other metal or mineral matter in such proportion that the gold, silver, platinum, or rarer metal or combination of them is or was of greater value than the baser metal or mineral in which it is or was contained. (Art. 2 of No. 34 of 1920 and of reg. of 1931).

¹⁶ "Precious stones" means rough and uncut stones only. (Art. 2 of No. 34 of 1920 and of reg. of 1931).

¹⁷ "Minerals" includes ores or compounds of minerals. (Art. 2 of No. 34 of 1920 and of reg. of 1931).

¹⁸ "Metals" includes ores or compounds of metals. (Art. 2 of No. 34 of 1920 and of reg. of 1931). (1754)

The Governor may grant to different persons, for different purposes, concessions, leases, or licenses with respect to the whole or a portion of the area included under a mining concession, lease, or license, provided that (1) the holder of the right for different purposes shall not interfere with the actual workings of the concessionaire, lessee, or licensee already in occupation and shall not be entitled to acquire any mining tenement upon the land except at a distance of more than 200 yards from those workings or from crushing, smelting, or other works; and provided that (2) a license to mine for precious stones shall not be granted with respect to an area held under a concession to mine, dredge, or wash for gold, except to the holder of the gold concession, and vice versa. (Arts. 16 and 32, No. 34 of 1920).

The Governor may grant to others than the concessionaire, lessee, or licensee the right to construct a pumping plant, hydroelectric plant, pipe line, transmission line, telegraph line, wireless or radio station, tramway, railway, and similar constructions, or to take, lead, or convey water from the land, provided that payment is made to the concessionaire, lessee, or licensee for all damage he may suffer - compensation to be determined, in default of agreement, by two arbitrators or their umpire, according to the provisions of the arbitration ordinance. (Arts. 20 and 33, No. 34 of 1920).

All metals and minerals obtained on a claim ¹⁹ without the permission of the owner thereof shall be his property, provided he is complying with the regulations as to the marking of boundaries and has paid the rent owed by him. (Arts. 30 and 33, No. 34 of 1920).

Plurality of applications. - In case more applications than one are received for a concession or lease of the same area, the Governor may direct that the matter be exposed to public competition at auction. However, the Governor may direct that the competition be restricted to any two or more of the applicants. (Art. 23, No. 34 of 1920).

Area. - No concession, lease, or license shall be granted for an area of more than 1,000 acres, except with the approval of the Secretary of State. (Art. 19, No. 34 of 1920, as amended by Art. 2 of No. 20 of 1931; art. 33, No. 34 of 1920).

3. Mining License for Private Land

With respect to licenses to mine on private land, article 52 of the mining ordinance, as amended by article 4 of ordinance No. 20 of 1931, reads as follows:

¹⁹ Claim includes the area of Crown land with respect to which a concession, lease, or license has been granted and includes any claim located, whether a concession or license has been issued or not, and includes the area of any land or water with respect to which a dredging concession is granted. (Art. 2, No. 34 of 1920, and art. 2, reg., 1931).

The Commissioner may, with the approval of the Governor, subject to the provisions of article 53 of this ordinance, issue a license to any person authorizing him to enter on private lands and there search and mine for, take, and appropriate gold, silver, valuable minerals, precious stones, or other metals or minerals, and in the case of private lands granted after the commencement of the mining (mineral oil) ordinance, 1910, mineral oil, asphalt, coal, or other substance of a like nature.

See section of this paper entitled "Ownership."

4. Dredging Concession

The Governor in Council may, on any conditions he considers meet, grant a concession or lease to anyone to dredge any river or creek or any portion thereof (including the banks and adjacent flats) and to take all metals and minerals, including gold, silver, precious stones, and valuable minerals. (Art. 17, No. 34 of 1920).

LESSEES AND TRIBUTORS

A lessee is a person to whom the holder of a claim license has sublet his claim or a part thereof. The holder must obtain written permission from the Commissioner or warden in order to sublet his claim. (Art. 130, reg., 1931). A tributor is the holder of a mining permission, called a mining privilege, which must be obtained before either the holder of a license or the lessee of a claim may allow anyone (other than a person duly registered to serve as a laborer) to work on the claim on any terms or in any capacity. (Arts. 131-1 and 131-2, reg., 1931).

The main provisions with respect to lessees and tributors, covered by articles 135 to 143 of the regulations of 1931, are given in the succeeding paragraphs.

Obligation of holder of license and of lessee. - The holder of a license, notwithstanding any subletting or permission, is subject to all the obligations of the mining law and regulations. (Art. 131-3, reg., 1931).

Every lessee (except as the regulations provide otherwise) shall (without prejudice to the liability of the license holder) be deemed to be a joint holder of such claims with respect to all obligations imposed on holders of licenses. (Art. 131-4, reg., 1931).

Application for mining privilege. - A prospective tributor shall make application (in the prescribed form) for a mining privilege to the Commissioner or the warden, who may refuse to issue a mining privilege to any person convicted of an offense against section 20 of the employers and servants ordinance or any ordinance amending it. Appeal from a decision of the Commissioner or warden may be taken to the Governor, whose decision is final. (Arts. 132-1 and 132-3, reg., 1931).

Endorsement of privilege. - The person in charge of a claim shall endorse a mining privilege before permitting a tributor to work on the claim. The privilege after endorsement becomes a contract between the license holder (or the lessee) and the tributor, making it a breach of the mining regulations for the one in charge to refuse to allow the tributor to complete the work for which he secured the permission or for the tributor to refuse to complete any task agreed upon. (Art. 135, reg., 1931).

Duration of privilege. - No mining privilege shall be for a period of more than 12 months. (Art. 133, reg., 1931).

EASEMENTS

A person may hold on any watercourse that is not navigable claims extending from the point where it ceases to be navigable to its source, navigability to be determined by the Commissioner or the warden. No claim other than a dredging concession shall include the bed of a navigable river or watercourse. (Art. 55 and 56, reg., 1931).

Every holder of a claim is entitled to the free use of water naturally passing through or near his claim and not already lawfully appropriated, subject to the approval of the Commissioner or the warden. (Art. 58, reg., 1931).

The natural channel of a river or creek is considered to be a public tailrace, into which every claim holder shall be entitled or obliged by a warden to turn the tail water at the end of the claim. (Art. 64, reg., 1931).

The full text with respect to water rights is found in part 8, articles 55 to 67 of the regulations of 1931.

Other easements described in the 1920 law or the 1931 regulations include timber rights, way rights, and mill and other building sites. (Arts. 21 and 33, No. 34 of 1920; arts. 62-2, 63, 68-1, 70-1-2, and 71, reg., 1931).

All fees, rents, royalties, duties on profits, and other moneys payable under the mining law or regulations shall be paid over to the Colonial Treasurer, for the use of the colony, as long as the Legislature makes provision, to the satisfaction of His Majesty, for maintaining the civil-list establishment of the colony. (Art. 106, No. 34 of 1920).

Fees

The table of fees, as given in the second schedule of the mining regulations of 1931, is as follows:

For a prospecting license	\$5.00
For filing a notice of location of any claim and application for a license48
For filing an application for a concession or lease	10.00

For a certified copy of particulars relating to a prospecting license12
For a license to mine for gold for each financial year or part thereof	5.00
For a license to mine for precious stones for each financial year or part thereof	14.00
For a license to mine for gold and precious stones for each financial year or part thereof	19.00
For a license to mine for valuable minerals for each financial year or part thereof	10.00
For every duplicate license issued50
For every duplicate concession or lease issued	1.00
For transfer of each claim license	4.00
For every mining privilege24
For each certificate, including registration of labor24

Surveys:

For the survey of any area comprised within the boundaries of a tract to be held under a claim license, concession, or lease, the applicant shall deposit the estimated cost of the survey and where the actual cost exceeds the amount deposited shall pay the excess cost after survey, and similarly where the actual cost is less than the amount deposited, shall be entitled to a refund of the difference.

Fees in proceedings before the Commissioner or warden:

Filing complaint48
Summons of a witness24
Copy of evidence or any document, per page of 18 lines12

Fees under the tax ordinance for 1926 ²⁰ were:

1. For a license to trade in or purchase gold	25.00
2. For a license to purchase, sell, or export rough or uncut precious stones	100.00

Rent

Amount. - Article 27 of the regulations of 1931 stipulates that rent, which is payable in advance, shall be that specified in the second schedule of the regulations. The second schedule, however, does not contain rents designated as such. The Imperial Institute ²¹ makes the following explanation:

²⁰ Imperial Institute (British), Work cited, p. 36. The tax ordinances vary annually.

²¹ Imperial Institute (British), Work cited, p. 36.

The second schedule contains a table of fees, the fee for a license to mine for gold being \$5, that for a license to mine for precious stones \$14, and that to mine for gold and precious stones \$19 for each financial year. Of course, a wide difference exists between a fee payable for a license and rent. It would seem, however, that the intention of the regulations is to treat these annual payments as rent, with all the consequential rights and obligations, including the right set forth in article 26 to sell claims for nonpayment of rent.

Subsection 3 (c) of article 35 and subsection 6 of article 37, regulations of 1931, provide as follows:

The annual rent under an exclusive prospecting permit shall be $7\frac{1}{2}$ cents an acre during the first 3 years the permit is in force. If the permit is renewed or extended, the rent shall be increased to 15 cents an acre for the fourth year and to 25 cents for each succeeding year.

According to the Imperial Institute, ²² an order in council of May 15, 1923, fixed the following rents:

With respect to leases of Crown lands, the rental is 20 cents an acre a year where the rights conferred are the rights to mine, drill, or wash for gold, silver, and valuable minerals, and is 50 cents where the rights relate to precious stones, and is 70 cents where the rights relate to gold, silver, valuable minerals, and precious stones.

In the case of concessions or leases for dredging, a yearly rental of 10 cents an acre a year is payable where the right is to dredge for gold, silver, and valuable minerals, and 20 cents an acre a year where the right is to dredge for gold, silver, valuable minerals, and precious stones.

The Governor in Council may reduce a rent ordinarily payable in a special case where organized geological and mineralogical surveys or diamond drilling or other tests of a scientific or technical nature are to be undertaken, the results of which would be advantageous to the colony. (Art. 36, reg., 1931).

Mode of payment. - In connection with an exclusive permit, the first instalment of annual rent is due (upon notification of the approval of the application) for the period between the date upon which notification of approval is given and June 30 or December 31 next; if this rent is not paid within 1 month, the application may be abandoned. Thereafter the rent is due in two equal half-yearly payments, in advance, on January 2 and July 1. (Art. 35-3-c, reg., 1931).

Likewise, if the applicant for a mining concession or lease does not pay the first year's rent in advance or within 3 months of the notification of the approval of his application, the application shall be declared void. (Art. 43-6, reg., 1931).

²² Imperial Institute (British), Work cited, p. 36.

Penalties for nonpayment of rent. - The Commissioner shall, in the month of March of every year, for three successive Saturdays, advertise for sale the rights of the licensees in all claims for which rent has not been paid for the current year. The holder of a license may at any time before the sale pay the rent due, plus 50 cents for the cost of advertisement of each claim. (Art. 26, reg., 1931).

The Commissioner or a warden may detain and sell any gold or precious stones obtained from any claim, in case the rent is not paid, and may deduct the proceeds of the sale from the amount of rent due to the Government. (Art. 27, reg., 1931).

Royalties, or Duties on Profits

The holder of a mining concession, lease, or license shall pay on the value of all metals or minerals (including gold and silver and their ores and compounds, precious stones, mineral oil, asphalt, coal, and like substances) found and appropriated on his area the royalty, or duty on profits, prescribed in any ordinance or regulations in force at the time the right is granted or in any subsequent ordinance or regulations if so provided in the concession, lease, or license. (Arts. 22 and 33, No. 34 of 1920).

The Governor in Council may grant any concession or lease upon the terms that no royalty shall be paid with respect to any metals or minerals obtained thereunder, if provision is made to the satisfaction of the Governor in Council for the payment of any sums of money or other consideration in lieu of royalty. (Art. 26, No. 34 of 1920).

In lieu of royalty, a duty shall be paid of 5 percent per annum on the annual amount of all profits made through a concession, lease, or license for a claim on which gold, tin, or copper is obtained by crushing ore with machinery (consisting of a mill containing at least 10 stamps or other mill equivalent to a battery of 10 stamps) or for a claim on which gold or tin is won by means of dredging machinery. All sums payable under this provision, as well as all penalties and other charges, shall be a first charge on the claim and on the machinery, mills, dredges, or other assets connected with the claim and shall be recoverable by parate execution. (Art. 35, No. 34 of 1920).

Articles 35 to 51 (part 4) of the consolidated mining law (No. 34 of 1920) contain numerous other provisions with respect to duty on profits, among which are the following:

Assessment. - The commissioner (who may require verification of all statements of profit) shall assess the amount of duty. No deductions shall be allowed with respect to capital expenditure or for losses disconnected with the business or for debts (except bad debts) or for any expenses not made exclusively for the purposes of the industrial or trading concern, as distinct from domestic or private purposes. Any one aggrieved by an assessment may appeal to the Supreme Court within 6 weeks of the notice of assessment. (Arts. 36 and 40 to 42, No. 34 of 1920).

Annual statement. - A true and correct statement of profits during the 12 preceding months must be prepared and delivered to the Commissioner within 3 months after the date upon which the accounts of the trade or business have been made up. (Art. 37, No. 34 of 1920).

Royalties, as fixed by articles 126 to 129 of the mining regulations of 1931, are as follows:

1. For gold 50 cents an ounce and for silver 4 cents an ounce, or such amount as the Governor in Council may from time to time direct.
2. For precious stones 50 cents a carat, or such amount as the Governor in Council may determine.
3. For valuable minerals, minerals or metals, their ores and compounds to which the regulations refer, other than gold, silver, and precious stones, such amounts as the Governor in Council may from time to time determine.

Notwithstanding anything in the regulations, royalty, or duty on profits, as the case may be, shall be payable on all gold, and royalty shall be payable on all silver and precious stones, valuable minerals, minerals or metals, their ores and compounds found and appropriated under any concession lease, or license, whether declared in accordance with the provisions of the mining regulations or not. If they are lost or stolen, or if from any cause royalty or duty on profits cannot be paid in conformity with the regulations, the Colonial Treasury may recover such royalty or duty on profits by parate execution. (Art. 129, reg., 1931).

COMPANIES INCORPORATED OUTSIDE THE COLONY

According to part 9, article 248, of the companies (consolidated) ordinance, No. 17 of 1913, no company other than a company incorporated in the United Kingdom or in a British colony shall be entitled to hold lands in the colony of British Guiana, except upon the filing of the documents and particulars set forth in article 247 of the same ordinance.

Section 2 of article 247 reads as follows:

Every company incorporated outside the colony that after the commencement of this ordinance establishes a place of business (which includes a share-transfer or share-registration office) within the colony shall, within 3 months after the establishment, file with the registrar,--

1. A certified copy of its charter, statute, or memorandum and articles, or other instrument declaring or defining its constitution, and, if the instrument is not written in the English language, a certified translation thereof.
2. A list of its directors.

3. The names and addresses of some one or more persons resident in the colony that are authorized to accept on its behalf service of process and notices required to be served upon it.

If any alteration is made in any of the instruments, or the directorate, or the names and addresses of any of the persons, aforesaid, the company shall within 3 months file with the registrar a notice thereof.

Every company to which this section applies that uses the word "limited" as part of its name shall state the name of the country in which it is incorporated in every prospectus inviting subscriptions, in every place where it carries on business, and in all billheads, letter paper, notices, advertisements, and other official publications. (Sec. 4, art. 247).

A fee of \$10, or such other fee as the Governor in Council may direct, shall be paid for registration of any document mentioned in the foregoing paragraphs. (Sec. 5, art. 247).

A fine in a sum not to exceed \$240 is imposed for failure to comply with any of the requirements of section 247 and one of \$24 for every day of continued default. (Sec. 6, art. 247).

MINING PARTNERSHIPS

Part 6 of the consolidated mining law does not exclude the application of the general law of partnerships; but it does amplify the general law to include among mining partnerships associations that would not otherwise be so considered. The provisions, therefore, of the partnership law of 1900 may be invoked; and, further, the provisions of part 6 of the mining law may be ignored, -- article 36 thereof stipulating that by written contract the application of the provisions relating to mining partnerships may be limited or excluded.

Article 63 provides also that the provisions of the consolidated law do not apply to any company or association incorporated or registered under the repealed companies ordinance of 1898 or under the companies consolidated ordinance (or amendment thereto).

A mining partnership exists when one or more persons own or acquire a claim for the purpose of working it and are actually engaged in working it or jointly employing others to do so, whether a written contract of partnership exists or not. An express agreement to become partners or to share profits and losses is not necessary. (Art. 55, No. 34 of 1920).

Further details with respect to mining partnerships are found in articles 55 to 63, inclusive, No. 34 of 1920.

REGULATION OF MINES

As little or no deep mining is done in British Guiana, the mining being chiefly of the claim type for gold, diamonds, or bauxite, elaborate regulations for mine management have not been formulated.²³ By article 111 of the regulations of 1931, the Commissioner is given the power to declare any claim or part thereof to be a mine or to alter or revoke any declaration, thus bringing all mining under the provisions covering actual mining operations. Some of the more important subjects of the regulations are the following:

Management and inspection (arts. 112, 120, and 123, reg., 1931)

Sanitary provisions (arts. 101 to 110, reg., 1931)

Safety provisions (arts. 117, 118, 120, 121, 122, 124, 125, and 201, reg., 1931)

Labor provisions (arts. 113, 114, 115, and 144 to 168, reg., 1931)

Plans (art. 119, reg., 1931)

In this connection, there should be consulted also the employers' and laborers' ordinance, 1909 (as modified by the amendment ordinance of 1914), and the accidental death and workmen's injuries ordinance, 1916.

SPECIAL PROVISIONS WITH RESPECT TO GOLD, VALUABLE MINERALS,
AND PRECIOUS STONES

Sale and Purchase

Both the seller and the purchaser of gold, valuable minerals, and precious stones must hold licenses to trade, issued in accordance with the tax ordinance for the time being in force, with the following exceptions:

1. The holder of a license to trade or the holder or licensee of a claim may sell small quantities of gold, etc., to others than a licensed trader for the personal use only of the purchaser.
2. Any holder or lessee of a claim or his duly authorized agent or any holder of a prospecting license that has obtained gold, etc., on his claim during prospecting may sell to anyone holding a purchasing license.
3. Any tributor may sell any gold, etc., obtained by him from a claim on which he is working under a contract that entitles him to all or part of the products won by him, to anyone holding a license to purchase in a mining district. (Arts. 181 and 185, reg., 1931).

No gold, valuable minerals, or precious stones may be sold or purchased except on the premises licensed for trade therein. (Art. 187, reg., 1931).

²³ Imperial Institute (British), Work cited, p. 43.

Further details are covered by articles 181 to 187, mining regulations of 1931.

Export of Precious Stones

No one without the written authority of the Governor shall export precious stones won by him or on his claim or purchased by him under a license to trade, except through a bank engaged in business in Georgetown. Every bank, when so required by the Commissioner, shall furnish particulars with respect to all such exports. (Art. 188, reg., 1931).

Details with respect to the declaration of precious stones before their export, checking, weighing, sealing, and stamping by the Department of Mines, together with the penalty for a breach of the regulations, are found in articles 189 to 193 of the regulations of 1931.

Miscellaneous

The strict control of gold, valuable minerals, and precious stones is further indicated by the following listing of provisions of the law: Prevention of theft (arts. 64 to 75, No. 34 of 1920); records 24 (arts. 169 to 171, reg., 1931); conveyance 25 (arts. 173 and 174, reg., 1931); determination and payment of royalty (arts. 175 to 179, reg., 1931); detention (art. 179, reg., 1931); and disposal of forfeited product (art. 200, reg., 1931; art. 106, No. 34 of 1920).

PART III. - BAUXITE

GENERAL

All citations in this section refer to the law governing bauxite exploration and exploitation, the bauxite mining regulations, 1930.

Bauxite, in the meaning of the law, includes "as well as the mineral so named every other mineral of like nature that in the opinion of the Commissioner is of equal merchantable value or quality." (Arts. 1 and 2).

The instruments under which bauxite is searched for and mined are (1) an exclusive permission to explore and (2) a mining lease. It is assumed that a permission must precede a lease. Article 21 reads:

The holder of a permission . . . may, if he has complied with the provisions of the regulations and of his permission, and if the Governor is satisfied that the exploration work done has led to the discovery of bauxite deposits, have the right to obtain a lease

24. Applying likewise to tin and copper (art. 49, No. 34 of 1920).

25. Applying likewise to tin and copper (art. 48, No. 34 of 1920).

A prospecting permit (and, by assumption, a mining lease) may not be issued (or transferred) to any resident outside of the colony unless such person is represented by a duly constituted attorney resident in the colony and the power of attorney is deposited in the deeds registry and a certified copy thereof is filed with the Commissioner. (Art. 10).

PROSPECTING RIGHTS

Exclusive Prospecting Permission

General. - A permission gives the right to the permittee to occupy and explore any particular area of Crown land only for the purpose of testing the value thereof for bauxite. (Art. 3-1). It shall not confer any right, title, or interest in the soil or in any gold, silver, valuable minerals, precious stones, mineral coal or mineral oil or substance of like nature, or in any timber or balata or other substances obtainable therefrom. It shall confer only such right in the bauxite as is provided in the permission, and it shall not entitle the holder to mine or ship bauxite (except samples). (Art. 13 and sec. 8, permission form, No. 2, sched. 1).

The granting or the refusing of a permission is at the absolute discretion of the Governor in Council (subject to the approval of the Secretary of State). The Governor may, if more than one application shall have been filed, direct that a permission be competed for at auction between applicants. (Art. 6).

Duration. - A permission shall not be issued for more than 1 year, but it may be renewed for two or more successive periods of one year each, provided that the Governor in Council is satisfied that sufficient exploration work has been done to warrant such extension or that good reason exists for its not having been done. (Art. 15). If the Governor is not satisfied at the end of any year with respect to the work done, he may nevertheless extend the permission for 1 year upon the applicant's depositing with the Commissioner a sum of not less than \$1,000 or more than \$5,000, as the Governor may decide, or upon his entering a bond, with one or more sureties to the satisfaction of the Commissioner, as a guarantee that sufficient work will be carried out under the renewed permission. If the Commissioner deems the work carried out to be sufficient, the money deposited shall be refunded; if he deems it to be insufficient, all or whatever part of the deposit the Governor in Council shall direct shall be forfeited. (Art. 16).

Rent and provisional permission. - An annual rent of 1 cent for every acre shall be payable in two half-yearly installments, in advance, on January 1 and July 1, - the first installment to be payable for the period between the date of the notification of approval of the application and June 30 or December 31 next, as the case may be. The applicant is allowed 3 months from the date of the notification in which to pay the first installment and within the 3 months is considered to be under a provisional permission. If the rent is not paid within the 3-month period, the Commissioner shall advertise the area as abandoned. (Arts. 7 to 9).

Work requirements. - The Commissioner may at any period during which a permission is in existence call upon the holder to furnish a true and proper statement of the nature and extent of the exploration work carried out within any specified period and the cost thereof, verified by a statutory statement if it is so required. If such statement is found to be wilfully misleading, the permission may be cancelled, and any deposit may be forfeited, without prejudice to any further action that may be taken. (Art. 17).

A permittee shall within 6 months from the date of the permission commence and thereafter diligently continue to search for bauxite. (Sec. 4 of permit form, No. 2, sched. 1).

Records of exploration work. - The holder of a permission shall at the conclusion of operations transmit to the Commissioner full information and such records of his search (in the way of a map showing the location of the bauxite deposits, the area, and a report thereon), together with such samples as may be called for, as will satisfy the Governor that bona fide exploration has been done, before a mining lease will be considered. (Art. 20).

MINING RIGHTS

Mining Lease

The rights conferred by a mining lease, as given in article 21 and in the preamble of the mining lease form (No. 3 of schedule 1) are as follows:

To hold and enjoy the said premises . . . for the purpose of winning, working, digging, mining, raising, converting, manufacturing,²⁶ and carrying away all bauxite that may be found therein, with the right to carry on upon the said land all operations incidental to or connected therewith, including the right to erect on the said land such dwellings or erections as may be necessary for the purposes aforesaid and the right to erect thereon and on any intervening Crown and colony lands (but as to such intervening lands only in situations to be first approved of in writing by the lessor) such aerial highways, rail lines, and other means of communication as may be required in order to connect the lessee's workings on the said lands with navigable waterways nearest thereto or with the nearest railway.

A lessee shall have no right to take or extract any gold or silver or other metals or any minerals or precious stones or mineral oil, timber, or other substances or thing, except as provided in the lease; but nothing shall prevent the lessee from using or removing such materials as sand, clay, rock, laterite, and like substances incidental to or connected with his mining for bauxite. (Preamble to lease form, No. 3, sched. 1).

Application. - An application for a lease, which shall be made to the Commissioner, must be preceded by a report upon the exploration work (referred to in a preceding paragraph) of the holder of a permission, or the application will not be considered. If the prescribed filing fee is paid,

26. Manufacture means the process of extracting aluminum or any other metal from bauxite by any means whatever. (Art. 2).

if the permit holder has complied with the provisions of his permission and with those of the bauxite mining regulations, and if the Governor is satisfied that the exploration work has lead to the discovery of bauxite deposits, the Governor may grant the petition for a lease, provided that he may refuse to grant plots in a shape that he may consider inconvenient, in which case the applicant shall be entitled to revise his selection within the limits of the areas covered by his permission. (Art. 21).

Rent. - An annual rent of 20 cents an acre shall be payable in advance, without demand, from the date of the expiration of the permission or from the date of the lease (whichever shall be earlier). (Art. 22).

Royalty. - Royalty is payable on all bauxite won from Crown lands whether exported from the colony or used for manufacture in the colony whenever it is exported or used. The rate for exported bauxite is 10 cents a ton (2,240 pounds); the rate for that used for manufacture in the colony is 2 cents a ton. (Art. 25).

A lessee shall pay a minimum annual royalty charge equivalent to the royalty that would be payable if 5 tons of bauxite were exported for each leased acre, provided that a lessee shall pay only a minimum royalty charge equal to the royalty on 1, 2, 3, and 4 tons for each acre for the first, second, third, and fourth years of the term, respectively. Such minimum royalty charge may be averaged over 5-year periods. (Art. 26).

TRANSFER

A lease shall not, without the written consent of the Governor, be transferred (art. 29); and in case it contains a condition forbidding transfer, neither a lease nor a permission shall be transferred under any circumstances (art. 19).

The procedure to be followed with respect to an intended transfer is outlined in article 19.

PREEMPTION OF BAUXITE IN TIME OF WAR OR OTHER NATIONAL EMERGENCY

In time of war or in any national emergency (of which the Governor shall be the sole judge), the Governor shall have the right to preempt all bauxite or products thereof, wherever they may be, at a price to be agreed upon between the lessee and the Governor; and the lessee shall, to the best of his ability, supply, when so requested, such quantity of bauxite and products thereof as may be required. If no price is agreed upon, it shall be the commercial value of the product at the place where it shall be supplied, to be determined by two arbitrators, one to be chosen by the lessor and one by the lessee, with power to appoint an umpire, -- such arbitration to be held under the arbitration ordinance of 1916 or of any modification or reenactment thereof. If the amounts and the times of delivery are in excess of the total capacity of the lessee's plant, the arbitrators shall fix the price so as to compensate the lessee for loss suffered in being required to increase the capacity of the plant; but no allowance shall be made if the joint demand from

the land under the lease and other lands to be leased under Permissions that may be granted to the lessee does not exceed 10,000 tons a year. (Plant includes equipment and necessary facilities). (Sec. 17, mining lease form, No.3, sched. 1).

The lessee shall not, without the written permission of the Governor or his representative, supply to any alien, either directly or indirectly, any bauxite from the land or the resulting products thereof (whether in the colony or not).

The Governor may prohibit, absolutely or under any conditions or restrictions he thinks fit, the removal of any bauxite or products thereof from the colony. (Sec. 17, mining lease form, No. 3, sched. 1).

EXPORT OF BAUXITE

Every person desiring to export bauxite shall give to the Comptroller, in addition to the information ordinarily required by the customs laws and regulations (art. 27):

1. The situation of the land from which he has won and removed the bauxite and a statement as to whether the land is Crown or colony land or is privately owned.
2. Such particulars of the title or ownership rights under which he has won and removed the bauxite as will prove that the person desiring to export is lawfully entitled so to do.

In the case of bauxite declared to have come from private property, the Comptroller may require a certificate from the Commissioner to the effect that the bauxite in question was won on privately owned property granted before the passing of the mining ordinance of 1903 and that it is not subject to the payment of royalty. (Art. 28).

FEES

The table of fees, as given in the second schedule of the bauxite regulations, 1930, is as follows:

For filing an application for an exclusive permission or a lease	\$10.00
For filing any notice of intended transfer of an exclusive permission	4.00
For a certified copy of an exclusive permission or a lease (per page)50
For a certified copy of a diagram attached to an exclusive permission or lease	2.00
For a compass survey of the boundaries of any area for which an exclusive permission is applied (exclusive of the cost of cutting boundary lines) -	
For the first 500 acres	50.00

For every additional acre up to 5,000 acres .	.02
For every additional acre over 5,000 acres .	.01
For a precise survey of the boundaries of any area applied for under a lease (including the cost of cutting boundary lines) --	
For the first 50 acres or part thereof . . .	30.00
For every additional acre over 50 and up to 500 acres30
For every additional acre over 500 acres and up to 1,000 acres20
For every additional acre over 1,000 acres . .	.10

Provided that where the actual cost of surveying any tract above 1,000 acres exceeds the amount deposited, the applicant shall pay the excess cost after survey; and, similarly, where the actual cost is less than the amount deposited, the applicant shall be entitled to a refund of the difference. The "actual cost of survey" shall include the surveyor's salary and allowances.

PART IV. - OIL AND NATURAL GAS

GENERAL

Draft regulations, to be known as the mining (oil and natural gas) regulations, 1931, to repeal the regulations of 1912,²⁷ were submitted for the approval of the Secretary of the Colonies on January 30, 1931, and were published in the Official Gazette of British Guiana of January 31, 1931. Up to the time of the writing of this paper, the proposed regulations had not been made law. (See section of this paper entitled "Rights of Foreigners.") Nevertheless, it has been thought advisable to mention the principal features of this legislation, - that is, of the law itself, - many of the essential provisions being embodied in the forms for licenses and leases, which may be changed at any time by executive order.

The substances included under the provisions of these regulations are oil and other bituminous minerals ²⁸ (except coal) and natural gas.²⁹ (Arts. 2, 27, 29, and 32).

The instruments by which rights are granted under this act are (1) exploration licenses, (2) prospecting licenses, and (3) mining leases. The exploration license does not necessarily precede the prospecting license, but a prospecting license must precede a mining lease; in fact, the chief differ-

²⁷ Continued in force by article 110 (4) of the mining law of 1920.

²⁸ Bituminous minerals shall be deemed not to include shale from which oil can be obtained by destructive distillation.

²⁹ Natural gas means any gas obtained from borings and consisting primarily of hydrocarbons.

ence between a prospecting license and a mining lease is in its tenure, - the former being granted for not more than 2 years and the latter for 21 years.

An exploration license, except to the grantee, lessee, or licensee of agricultural land, shall not be issued except for Crown land demarcated on an "Official Exploration Map," which shall be open to inspection in the Department of Mines. A prospecting license may be granted to the holder of an exploration license, to the grantee, lessee, or licensee of agricultural land, or to any other applicant fulfilling the obligations of the law. (Arts. 3, 23, and 25).

Applications for licenses and leases and renewals thereof, made through the Commissioner of Mines, are granted by the Governor in Council with the approval of the Secretary of State, except with respect to an exploring license, which may be granted by the Governor in Council. (Arts. 6, 7, 23, and 33).

NATURE OF RIGHTS

An exploration license gives the licensee an exclusive right to explore and search for oil (or other bituminous minerals, except coal) and/or natural gas in a given area (art. 6); a prospecting license gives the holder thereof an exclusive right to mine, bore, search for, win, and work these mineral substances and to export the produce (art. 27); and the lease is a continuation of the prospecting license (art. 32).

DURATION OF LICENSES AND LEASES

An exploration license is issued for 1 year, with the right of renewal if the holder is able to prove that he has done real exploration work, the renewal license to be cancelled if within 6 months the licensee has not undertaken exploration work to the satisfaction of the Governor in Council. (Arts. 6, 16, 17, and 18).

A prospecting license is granted for such period as the Governor in Council may see fit, not in excess of 2 years (art. 28); it may be renewed for another period not to exceed 2 years (art. 30).

A mining lease shall be for a term of 21 years or for such longer period as the Secretary of State may approve; it may be renewed for a further period of 21 years or longer. (Arts. 33 and 34). Renewal shall be subject, of course, to the payment of all rents and royalties and the observance of all covenants of the lease up to the expiration of the term granted. (Art. 34).

AREAS

Under an exploration license, the Governor may grant to one applicant more than one block of "50 acres that shall not constitute one area." (Arts. 3 and 7).

Under a prospecting license, "the area shall be computed in square miles not to exceed 10 and not to be less than 1." Where practicable, the area

shall be laid out in a rectangular block. (Art. 27). A lease shall not cover an area greater than that covered by the prospecting license. (Art. 32).

The position and bearings of boundaries on the ground of all blocks shall be decided by the coordinates and bearings deduced by the Commissioner from the official exploration map. A sea or river shall be the boundary on the side abutting thereon. Beds of navigable streams shall not be included in an exploration license. (Art. 21).

FEEs, RENT, OR ROYALTY

The fee for filing an application for an exploration license shall be \$10. (Art. 5). The fee for the license or the renewal thereof shall be at the rate of \$50 for each block. It shall be paid within 1 month of the first advertisement (made by the Commissioner when decision has been reached with respect to granting the application), or the application shall be deemed to be withdrawn. (Arts. 8 and 16). The fees shall be returned in case the application, after objections have been entered (arts. 10 to 13), is refused. (Art. 14).

The fee for filing an application for a prospecting license is \$10. (Art. 26). For any prospecting area, an annual fee of 10 cents an acre shall be paid, subject to a minimum of \$64. All fees shall be paid in advance; and if the first fee is not paid within 2 months of the applicant's notice of approval of his petition, the application shall be deemed to have been withdrawn. (Art. 29).

Holders of mining leases shall be subject to such rent or royalty as the Governor in Council may approve. (Art. 33). A renewal of the lease shall be at the same surface rent as the original lease, but subject to the payment by the lessee of such yearly rent and royalty as may be then fixed by the Governor, provided that any increase in rent shall not exceed 50 percent of the rent reserved in the lease. (Art. 34).

The Government may waive or reduce fees ordinarily payable in cases where geological surveys or other scientific tests made by the licensee would be advantageous to the Government. (Art. 24).

TRANSFERS

A license or lease shall not be transferred or assigned without the previous written permission of the Governor. (Art. 38).

Approval of a transfer shall not be given unless it is proved to the satisfaction of the Governor in Council and the Secretary of State that the proposed transferee or assignee has the financial resources and technical assistance necessary for carrying on the contemplated work. (Art. 39).

UNITED STATES BUREAU OF MINES
SCOTT TURNER, DIRECTOR

INFORMATION CIRCULAR

MINING LAWS OF BRITISH HONDURAS



BY

E. P. YOUNGMAN

INFORMATION CIRCULAR

UNITED STATES BUREAU OF MINES

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By E. P. Youngman²

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2 - Formerly of Rare metals and nonmetals division, U. S. Bureau of Mines.

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PART I. - GENERAL

PREFATORY NOTE

This paper is one of a series of digests of foreign mining legislation and court decisions that is being prepared in advance of a general report relative to the right of American citizens to explore for minerals and to own and operate mines in various foreign countries. This interpretation of the mining laws of British Honduras has been prepared from the best available information in Washington and is released subject to amplification or correction, if necessary, by United States foreign-service officers.

INTRODUCTION

The mining law governing all mineral substances other than oil is ordinance No. 7 of 1927 (Minerals Ordinance, 1927), passed by the Legislative Council on March 25, 1927. The accompanying regulations, No. 3 of 1932 (General Minerals Regulations, 1932), made under the authority of section 66 of the minerals ordinance, were made by the Governor in Council on December 23, 1931.

The law governing oil is ordinance No. 32 of 1920 (Oil Mines Ordinance, 1920), passed by the Legislative Council on August 9, 1920. The accompanying regulations are the "Oil Mining Regulations of January 5, 1921," made by the Governor in Council by virtue of section 6 of the oil mines ordinance.

Other laws to which reference has been made in this digest include:

(1) The Crown lands ordinance³, chapter 38 of the new edition of the consolidated laws of British Honduras, containing the ordinances in force on July 21, 1924; (2) an ordinance to regulate the status and capacity of aliens within the colony, chapter 114 of the consolidated laws of British Honduras, 1924; (3) the companies ordinance (part 10), chapter 203 of the consolidated laws, 1924; and (4) the recording of deeds ordinance, chapter 214 of the consolidated laws, 1924.

All privileges granted with respect to oil take preference over those granted with respect to other minerals, even to the extent that, if an applicant for, or the holder of, a license or lease under the oil mines ordinance covering land leased under the general mining law shall request the Governor to terminate the mining lease and shall satisfy him that its termination is necessary and shall give security for any compensation payable for loss or damage to the lessee whose lease is rendered void, the Governor may, by notice served on the lessee, terminate the general mining lease. (Sec. 54, No. 7 of 1927.)

3 - Sections 26 to 29 of the Crown lands ordinance, which were repealed by section 67 of the minerals ordinance, 1927.

Likewise, although other licensees and lessees of Crown lands may exercise all the liberties and powers granted by the Crown lands ordinance, or any amending ordinance, they are prohibited from interfering with the rights of licensees and lessees under the oil laws. (Sec. 7, No. 32 of 1920.)

RIGHTS OF FOREIGNERS

In the absence of any prohibition to the contrary, it is assumed that foreigners enjoy equal rights with natural-born British subjects with respect to prospecting for and mining all mineral substances except oil in British Honduras. However, aliens having no established place of business in the colony may be required to deposit a \$250 bond before receiving a prospecting license.

The oil law definitely rules against aliens, and every oil mining lease includes a stipulation that the lessee shall be and remain a British subject or a British company under British control.

Section 8 of the oil mining law (No. 32 of 1920) reads as follows:

Any person or persons holding any license or lease granted under the provisions of this ordinance shall at all times be and remain British subjects; and any company holding any such license or lease shall be a British company registered in some part of the British Empire and shall have its principal place of business within His Majesty's dominions; and the chairman and the managing director (if any) and the majority of the other directors of the company shall at all times be and remain British subjects; and further, neither the lessee or lessees nor any lands the subject matter of any lease shall at any time be or become, directly or indirectly, controlled or managed by any alien or aliens or by any company other than a British company, as hereinbefore provided; and the local general manager of any lessee or lessees and such percentage as the Governor may prescribe of the local staff employed shall at all times be and remain British subjects.

A foreign company, within one month of the establishment of its business, shall file with the Registrar a certified copy of the charter, statutes, or memorandum and articles of the company, together with other required data. It shall file also yearly financial returns, in the form of a balance sheet. A company incorporated in British territory, having filed the documents and particulars required, shall have the same right to hold lands in the colony as if it were a company incorporated under the companies ordinance.⁴

4 - Secs. 248 and 249, pt. 9, chap. 203, revised edition of the Consolidated Laws of British Honduras, 1924, vol. 2, pp. 1488-1490.

OWNERSHIP

The Crown lands ordinance⁵ (sec. 47) reserves to the sole benefit of the Crown all precious metals and minerals and oil, together with the right to exploit them, subject only to proper compensation for injury to or upon the surface.

The mining ordinance (No. 7 of 1927), governing all mineral substances other than oil, in section 3, reads:

The entire property in and control of all minerals in, under, or upon any lands in the colony, or in or under all rivers, streams, and water-courses throughout the colony is and shall be vested in the Crown "save insofar as such rights shall not have been alienated" by any grant made or title adverse to the Crown acquired prior to the commencement of this ordinance.

Notwithstanding anything in this or any other ordinance contained, no grant of Crown lands made after the commencement of this ordinance shall be deemed to confer upon the grantee any right to any mineral whatsoever in such Crown lands, and all such minerals shall remain the absolute property of the Crown.

The oil mining law (No. 32 of 1920) limits the authority of the Government to grant oil privileges to (1) lands held by the Crown for the benefit of the colony, including lands covered by water, (2) lands upon which the right to enter, inspect, search, dig for, and carry away mineral oil is reserved to the Crown, or (3) lands under lease from the Crown the right to enter whereon, etc., is reserved to the Crown. (Sec. 3, No. 32 of 1920). The oil mining regulation (of 1921) in sections 6, 7, and 8 give conditions with respect to land alienated subsequent to December 20, 1907.

With respect to oil, the law evidently does not apply to lands alienated prior to 1907; with respect to other minerals, the minerals ordinance is not so clear, although the possible exception of certain previously alienated lands may be inferred from section 3, quoted in a preceding paragraph.

The Government may withdraw land granted for prospecting and mining rights in case the land is required for public purposes--compensation to be paid only to a mining lessee that is paying surface rent and only for interference with ways, works, buildings, or plant, and not for interference with his prospecting or mining privilege. (Sec. 53, No. 7 of 1927.) Or the Government, because of the existence of a mining right or a mining lease, may revoke a location ticket or resume possession of lands occupied under a Crown lease or permit to occupy--the mining lessee to pay to the Government

5 - An Ordinance to Regulate the Management and Disposal of Crown Lands: New edition of the Consolidated Laws of British Honduras, 1924, containing the ordinances in force on July 21, 1924, chap. 38, vol. 1, London, 1924, pp. 532-545.

the amount of the compensation paid by it to the person whose privileges have been withdrawn. (Sec. 35, No. 7 of 1927.)

The Governor may take from land subject to prospecting or mining privileges any materials required for the construction of railways, roads, buildings, or other public works; or he may use, or authorize the use of, any road constructed by the holder of prospecting or mining privileges. (The Governor may, however, before authorizing the use of such a road, require that a person using it shall make a fair contribution to the cost of construction.) Sec. 53, No. 7 of 1927.)

CLASSIFICATION OF MINERALS

Mineral substances, with respect to the mining laws of British Honduras, are placed in the following classes (sec. 2, No. 7 of 1927; secs. 2 and 3, No. 32 of 1920):

1. Metalliferous minerals, including antimony, arsenic, bismuth, copper, cobalt, chromium, cadmium, iron, lead, manganese, mercury, molybdenum, nickel, tin, tungsten, uranium, zinc, and all others of a similar nature, together with all ores containing them and combinations of any of them with each other or with any other substance except those that occur in the form of precious minerals.

2. Carbonaceous minerals, including anthracites, asphalt, brown coal, bitumen, and its compounds, coal, graphite, lignite, and all substances of a similar nature or combination of any of them with each other or with any other substance.

3. Earthy minerals, including asbestos, barytes, gypsum, marble, mica, phosphates, potash, rock salt, talc, soda, sulphur, steatite, slate, and all other substances of similar nature.

4. Precious minerals, including precious stones; namely, amber, amethyst, beryl, cat's eye, chrysolite, diamond, emerald, garnet, opal, ruby, sapphire, turquoise, and all other substances of a similar nature.

5. Precious metals, including gold, iridium, silver, platinum, or ores containing them: Provided that argentiferous lead ores containing not more than 4 ounces of silver to a ton shall be excluded from this class.

6. Mineral oil. The oil mining law covers oil and other bituminous minerals; and bituminous mineral means any substance other than crude oil containing petroleum oil or any product thereof.

CLASSIFICATION OF LANDS

For the purposes of prospecting and mining for all minerals save mineral oils, land is classified as follows (sec. 3, No. 7, of 1927):

1. Precious minerals may be prospected for and mined in, under, or upon any land of the colony.

2. Coal may be prospected for and mined in, under, or upon any of the lands in the colony except such lands as were granted in fee simple by the Crown or such lands with respect to which titles adverse to the Crown were acquired prior to July 1, 1886.

3. All other minerals may be prospected for and mined in, under, or upon any of the lands in the colony except such lands as were granted in fee simple by the Crown or with respect to which titles adverse to the Crown were acquired prior to the commencement of the mineral ordinance of 1927.

RESERVED OR EXEMPTED MINERALS AND AREAS

The Governor in Council, by notice in the Gazette, may declare any area closed for reserved minerals indefinitely or for any specified period; and anyone other than the holder of an exclusive prospecting license prospecting in such an area shall be liable to a fine of \$500 or imprisonment with hard labor. (Sec. 3 and 5, No. 7 of 1927.)

The Governor in Council, by notice in the Gazette, likewise may prohibit prospecting for any specified reserved mineral; in which case a prospecting right or license shall not cover such mineral unless the right or license expressly includes it. (Sec. 7, No. 7 of 1927.)

A prospecting right, an exclusive prospecting license, or a mining right shall not include the right to prospect or mine unless specifically granted by the Governor --

1. Within any Government station or township or on or under any lands used for or appropriated to any public purposes.

2. On or under land occupied by a town or village or by a market or burial ground or on or under lands within 50 yards of any Government of public building or works, or public road, or tramway, or within 100 yards of any railway without the consent in writing of the Governor first obtained.

3. On or under land actually under cultivation without the consent of the owner or occupier of the land.

4. On or under private land or land subject to a Crown lease, location ticket, or permit to occupy, within 50 yards of any building erected thereon (not a Government building) without the consent of the occupier of the building. (Sec. 6, No. 7 of 1927.)

A mining lease does not authorize the lessee to occupy or mine on or under land occupied by a village or by a market or burial ground or on land within 50 yards of any Government building or works, or any market, burial ground, public road or tramway or within 100 yards of any railway without the consent in writing of the Governor in Council. (Sec. 32, No. 7 of 1937.)

MINING ADMINISTRATION

The Governor in Council is the final mining authority, and he may make rules and regulations for carrying into effect the purposes of the general mining ordinance. (Sec. 66). The approval of the Secretary of State is necessary for all regulations dealing with the following matters: (1) Fees, rents, and payments to be made for any lease, license, or right granted; (2) the size and shape of the areas over which mining leases may be granted; (3) working conditions to be applied to mining leases; (4) the amount of royalty payable to the Government and the form and manner in which royalty shall be collected and paid; and (5) imposition of export duties on any mineral except those which the exporter satisfies the Governor in Council will be smelted in the United Kingdom or in a British possession.

The more immediate authorities are the Government Inspector of Mines and the Surveyor General. With respect to minerals in general, applications are submitted through the Government Inspector of Mines, of the Mines Bureau; with respect to oil, applications are submitted to the Surveyor General.

Section 6 of the oil mining ordinance (No. 32 of 1920) authorizes the Governor in Council to make regulations with respect to: (a) Method of making applications for privileges and determining priority of applicants; (b) the form, term, and conditions of licenses and leases; (c) manner in which areas, etc., shall be ascertained, described, and marked out; (d) the form of bond (if any) to be entered by an applicant; (4) conditions to be observed and duties to be performed by holders of privileges; (f) fixing of fees, rents, and royalties; and (g) the granting of pipe-line licenses under agreement.

PART II. - MINERALS OTHER THAN OILPROSPECTING

Prospecting is done under prospecting rights and exclusive prospecting licenses, neither of which entitles the holder to prospect for specified reserved minerals unless the right or the license expressly provides for it. (Sec. 7, No. 7 of 1927.)

The minerals raised belong to the Crown; they shall not be retained by a prospector except upon an application approved by the Government Inspector of Mines, who must before approving an application satisfy himself that the prospector performed only the work necessary to test the mineral-bearing quality of the land. (Sec. 14, No. 7 of 1927.) The application and the written consent must state the kind and the quantity of the minerals to be retained; and no mineral shall be retained with respect to which royalty is not fixed in the regulations. (Secs. 17 and 18, No. 3 of 1932.)

Application.--An application for a prospecting right is made to the Government Inspector of Mines, who may grant such a right. (Sec. 5, No. 3 of 1932.) The application shall be made in and give the details required by the prescribed form. (Sec. 6, No. 3 of 1932.)

Qualifications of applicant or conditions of granting prospecting right.--An applicant for a prospecting right must be able to read and understand the ordinance, must be 18 years old, and must give satisfactory proof that he is financially capable of paying traveling and prospecting expenses and any compensation to which his right may make him liable.⁶ Except with the consent of the Governor in Council, no person is eligible who or whose present employer has been convicted of an offense against the ordinance, or who or whose present employer has held and forfeited mineral privileges through breach of their conditions. (Sec. 8, No. 7 of 1927.)

Prospecting Right

The holder of a prospecting right may prospect on any land except land closed to prospecting or land subject to an exclusive prospecting license, a mining right, or a mining lease. He may construct his camp upon unoccupied land and take timber, other than protected trees, and water from lake, river, stream, or watercourse for domestic and prospecting purposes. He may sink shafts or wells or dig trenches, provided (a) that he shall not divert water from any river, stream, or watercourse without the consent of the Government Inspector of Mines; (b) that with respect to a forest reserve he shall have given notice to the forest officer in charge and shall have complied with the conditions imposed by him; and (c) that with respect to private lands or lands occupied under a Crown lease, a location ticket, or an occupation permit he

6 - The law makes this condition absolute; the regulations (sec. 6, No. 3 of 1932) make it optional with the Government Inspector of Mines.

shall have given notice of his intention to the owner or the occupier and if so required shall have given security against damage of any kind. (Sec. 9, No. 7 of 1927.)

The holder of a prospecting right upon entering a district for the purpose of prospecting shall inform the District Commissioner of his arrival. (Sec. 85, No. 3 of 1932.)

Duration.--A prospecting right shall remain in force for one year, except in the case of a person in the employ of another, whose right shall cease when he leaves the service of his employer, if the employer so requests. (Sec. 8, No. 7 of 1927.)

Transfer.--A prospecting right shall not be transferable. (Sec. 8, No. 7 of 1927.)

Exclusive Prospecting License

An exclusive prospecting license, which the Governor in Council may approve or refuse at his discretion, is granted to a person that has himself or through someone in his employ prospected the area for which a license is sought. (Sec. 10, No. 7 of 1927.)

An exclusive prospecting license gives to the holder sole prospecting privileges upon the licensed area. In addition to the privileges conferred by a prospecting right, the license confers upon the licensee the privilege of employing the necessary number of persons and on and over unoccupied land within the licensed area of erecting and maintaining such machinery and plant and constructing such ways as may be necessary. (Sec. 12, No. 7 of 1927.)

Application.--An application for an exclusive prospecting license shall be submitted to the Government Inspector of Mines; it shall be in the form prescribed; and it shall be accompanied by a plan of the area, drawn to a scale of not less than 1/25,000, with the relative and approximate details set forth in section 9 of the regulations of 1932 (No. 3 of 1932.)

The Government inspector shall send one copy of the application to the Colonial Secretary and one to the Surveyor General. The Surveyor General and the Government Inspector of Mines shall each furnish to the Colonial Secretary a written report, which shall contain any objections found. The application, with these reports, shall be considered by the Advisory Committee (comprising the Attorney General and any other officer or officers the Governor may appoint), which shall report to the Governor, who, if no objection has been raised, may or may not authorize a survey of the area or any portion thereof. (See section of this paper entitled "Surveys, Boundaries, Beacons, Etc.") (Sec. 10, No. 3 of 1932.) The Government Inspector of Mines also shall have discretion to determine that no survey is necessary. (Sec. 184, No. 3 of 1932.)

Area.--An exclusive prospecting license shall be granted for an area not to exceed 16 square miles or to be less than 1 square mile when a full square mile is available, except in the case of an area on which precious minerals may have been found. (Sec. 10, No. 7 of 1927.)

The area shall be rectangular in shape, with a minimum width of not less than one third of the length: Provided that the Surveyor General may for good cause permit exceptions. (Sec. 7, No. 3 of 1932.)

Duration and renewal.--An exclusive prospecting license shall be granted for 1 year, subject to renewal for further terms of 1 year to a maximum of 3 years in the case of alluvial workings and of 6 years in the case of lode workings. The Governor may grant a renewal with respect to an alluvial working for a fourth year, if proof is available that the prospecting operations were stopped or hindered by circumstances beyond the control of the licensee. The Governor, with respect to a lode working, may direct that a renewal shall be allowed for a specified portion of the area only. (Sec. 10, No. 7 of 1927.)

Applications for renewal shall be made, through the inspector, at least one month before the expiration of the license. (Sec. 16, No. 3 of 1932.)

Working requirements.--A licensee shall continuously and adequately carry on bona fide prospecting, to the satisfaction of the Government Inspector of Mines, unless the Governor shall, upon application of the licensee and for good cause shown, suspend such working obligation. (Sec. 15, No. 7 of 1927.)

Transfer.--The holder of an exclusive prospecting license shall not transfer his license without the consent of the Governor in Council, signified by an endorsement thereon. The transferee shall be liable for all rents and obligations that may have accrued at the time of the transfer. (Sec. 13, No. 7 of 1927.)

MINING

Mining is carried on under a mining right (for placer ground) or under a mining lease. During the consideration of an application for either privilege, the Governor may authorize the applicant to mine the area applied for, upon such conditions as the Governor may think expedient, and he may withdraw his authorization at any time. (Sec. 18, No. 7 of 1927.)

The Governor may force the holder of a mining right to apply for a lease, by revoking the mining right, should the reports upon the mineral-bearing qualities of the land justify such action. (Sec. 23, No. 7 of 1927.)

The Governor may at any time require that the holder of a mining right or of a mining lease shall give security for the payment of royalties, in such sum as he may direct, either in cash or by bond with two sureties--no mineral to be disposed of or exported until such security is given. (Sec. 84, No. 3 of 1932.)

Mining Right

A mining right, which is a placer mining right, confers upon its holder the exclusive privilege of mining the alluvial reserved minerals specified and to take and dispose of them, subject to the payment of the prescribed royalties. In addition, it confers the privileges granted by a mining lease by section 31 of the mining law (No. 7 of 1927), for which see page 14 of this paper. (Sec. 20, No. 7 of 1927.)

A mining right may be issued, by the Governor in Council, to the holder of a prospecting right or of an exclusive prospecting license. (Sec. 19, No. 7 of 1927.)

Application.--The provisions governing an application for a mining right are the same as those governing an application for an exclusive prospecting license (see p. 10 of this paper). (Secs. 10 and 19, No. 7 of 1927.)

The application shall be accompanied by a plan in duplicate, drawn to a scale of not more than 7 chains to 1 inch, which shall indicate the position of the stream with respect to which the application is made and the temporary beacons marking the boundaries of the area. (Sec. 21, No. 3 of 1932.) (See section of this paper entitled "Surveys, Boundaries, Beacons, Etc.")

Applications are dealt with in the same way as are applications for exclusive prospecting licenses (see p. 10 of this paper): Provided that the Government Inspector, in reporting upon an application, shall state whether it should be granted "having regard to the fact that it is not intended that a mining right should be granted unless there is reason to believe that the mineral-bearing qualities of the land are not such as to justify an application for a mining lease"; and that no survey or only such survey as is necessary to measure the length of the course of the stream applied for shall be required. The Governor, after considering the report, together with the report of the Advisory Committee, may refuse or grant all or any part of the area sought. (Secs. 10 and 22, No. 3 of 1932.)

With respect to an application for a renewal of a mining right, see rules governing the renewal of an exclusive prospecting license (see p. 11 of this paper). (Secs. 16 and 24, No. 3 of 1932.)

Area.--A mining right shall not cover an area upon which minerals have been discovered in payable quantities as a lode formation. (Sec. 19, No. 7 of 1927.) A mining right shall be granted with respect to a stream and the land lying within 100 yards of the center of the stream; not more than 1 mile along the course of any stream shall be included within the area the subject of one mining right. (Sec. 19, No. 3 of 1932.)

Duration.--A mining right is issued for 1 year, with the right of renewal for further terms of 1 year each. (Sec. 19, No. 7 of 1927.)

Working requirements.--The holder of a mining right shall continuously carry on adequate mining operations, to the satisfaction of the Government Inspector of Mines; and if he is not personally resident upon the land, he shall have a responsible agent thereon. (Sec. 21, No. 7 of 1927.)

Ancillary rights.--The holder of a mining right may cut and use on the land occupied by him any tree necessary for his operations or for domestic purposes, provided that he shall not take any protected tree without the consent of the proper forest official and on payment of the fees and royalties prescribed. (Secs. 22 and 34, No. 7 of 1927.)

Transfer.--The holder of a mining right shall not transfer his right or privileges thereunder without the consent of the Governor in Council; and the transferee shall be liable for all rents and obligations accrued at the time of the transfer. (Secs. 13 and 22, No. 7 of 1927.)

Mining Lease

The holder of a mining right may be required by the Governor to take out a mining lease or leases with respect to all or part of the land for which the right is held, if the Governor is satisfied through reports of the Government Inspector of Mines that the mineral-bearing qualities of the land are such as to justify revoking the mining right, in which event, of course, the holder of the right shall have preference in obtaining a lease or leases, provided he shall take advantage of that privilege within 2 months of the revocation. The holder shall have preferential right also to a mining right with respect to any portion of the area for which application for a lease is not made within 12 months from the revocation. (Sec. 23, No. 7 of 1927.)

The Governor may grant a mining lease to the holder of a prospecting right or of an exclusive prospecting license that has carried on bona fide operations on the area applied for or to the holder of a mining right with respect to any portion of the area under the right or, subject to the provisions of the preceding paragraph, to any person with respect to any land included in the area of a mining right that has been revoked. (Sec. 24, No. 7 of 1927.)

The holder of an exclusive prospecting license or a mining right that has fulfilled all the conditions attached thereto shall be entitled to the grant of a mining lease to cover any mineral for which he was authorized to prospect or mine, with respect to any portion of the area included in the license or right, subject to the provision that the Governor may exclude any portion of the area that he thinks proper. (Sec. 24, No. 7 of 1927.)

The Governor in Council may offer and grant leases for any mineral with respect to which a notice prohibiting prospecting has been issued (under section 7 of the mining ordinance); with respect to any area (not included in the area of an exclusive prospecting license then in force) in which reserved minerals in apparently payable quantities have been discovered or are known to exist, if no application for a mining lease with respect to such

area has been made by a person to whom the lease could be granted (as indicated in the last two preceding paragraphs); any area in respect to which an application for a mining lease has been made, if the application has been refused or has been withdrawn; or any area with respect to which a mining lease has been granted, if such lease has been forfeited, surrendered, or expired. (Sec. 24, No. 7 of 1927.)

The Governor in Council may, in any individual case, permit the lessee to mine any minerals found on the leased area that may not be included in the lease. (Sec. 30, No. 3 of 1932.)

Any lease, in general, confers upon the holder the exclusive right to remove and dispose of the minerals specified in the lease, subject to the payment of royalties. (Sec. 30, No. 7 of 1927.) More specifically, insofar as it may be necessary, a lessee shall have on the land leased the following rights (sec. 31, No. 7 of 1927):

1. To make all necessary shafts.
2. To erect, construct, and maintain houses and buildings for his own use and that of his agents and servants.
3. To erect, construct, and maintain such engines, machinery, buildings, and workshops and other structures as may be necessary or convenient.
4. To deposit rubbish produced in mining.
5. To lay water pipes and to make watercourses and ponds, dams, and reservoirs, and to divert any water on or flowing through the land; provided that no such diversion shall be made until the consent in writing of the Government Inspector of Mines has been obtained.
6. To construct and maintain all such tramways, roads, communications, and conveniences as may be necessary.

Kinds of leases.--Mining leases, according to sections 25 and 30 of No. 3 of 1932, shall come under the following classification:

- Class A. Metalliferous minerals and precious metals lode leases.
- Class B. Metalliferous minerals and precious metals alluvial leases.
- Class C. Mica leases.
- Class D. Precious stones leases.
- Class E. Carbonaceous minerals leases.
- Class F. Earthy minerals leases.

A class A lease covers all minerals specified therein, whether lode or alluvial; a class B lease covers the alluvial minerals specified therein; a class C lease covers mica; classes D, E, and F leases cover the mineral or minerals specified in the lease.

Application.--With respect to the application for a mining lease, the procedure is practically the same as that provided for applications for exclusive prospecting licenses (see p. 10 of this paper). (Sec. 25, No. 3 of 1932.)

Area.--The limits placed upon the size of areas under mining leases are as follows (sec. 26, No. 3 of 1932):

Classes A and B: The area under a class A or a class B lease shall not be less than 3 acres or less than 8.26 acres when polygonal in shape (if such areas are available). The area under a class A lease shall not exceed 50 acres; that under a class B lease 800 acres.

Class C: The area under a class C lease shall not exceed 40 acres.

Classes D, E, and F: Areas under D, E, and F leases shall be as prescribed by regulations to be made.

Except with respect to an area that has been the subject of a mining right, the area under a mining lease shall be (a) rectangular, the width not to be less than 200 yards or one third of its length, whichever is the shorter, or (b) a polygon of not less than 4 nor more than 10 sides, in which case no point on a given side shall be less than 200 yards distant from any other nonadjacent side, provided that the Surveyor General may allow a departure from these provisions if it is desirable that a boundary conform with that of another area. (Sec. 27, No. 3 of 1932.)

All mining leases shall be bounded by vertical planes from the surface boundary lines downward to an unlimited depth. (Sec. 28, No. 3 of 1932.)

Before the expiration of the first 6 months of a mining lease, the lessee shall mark off such parts of his leased area as he intends to use immediately for any purpose in connection with his mining operations. An officer appointed for that purpose shall measure the marked lands and shall inform the lessee as to the rent payable and as to whether any rent is payable to any private owner or occupier. (Sec. 38, No. 3 of 1932.)

Ancillary rights.--A lessee may cut, take, or use any tree on the land included in his lease except a protected tree, for which the consent of the proper forest officer is necessary, and for which fees and royalties are payable (Sec. 34, No. 7 of 1927.)

A mining lessee shall not use or occupy for any purpose in connection with his mining operations land other than that marked off and measured by the appointed officer (see preceding section) without first notifying the

District Commissioner of the extent of land, the purpose for which he desires it, and the date from which he will occupy it and pay rent therefor. A mining lessee may be required, by the Governor in the case of Crown land and by the owner in the case of private land, to accept a lease for or a permit to occupy land within the area of the mining lease that he may desire in connection with his mining operations. The mining lessee shall not be entitled to receive rent from any person residing on, or occupying any portion of, the area under the mining lease, whether the lessee pays rent for surface rights or not. (Sec. 38, No. 3 of 1932.)

Duration and renewal.--The term of a mining lease, which is determined by the Governor in Council, may not exceed 21 years. It may be renewed once for a term not to exceed 21 years, upon the payment of the required fees, provided the lessee shall have carried on his work in a business-like manner and the lease shall not have been liable to forfeiture. (Sec. 27, No. 7 of 1927.)

Working requirements.--The following requirements with respect to work are laid down, subject to modification by the Government Inspector of Mines or by the Governor in Council (sec. 38, No. 3 of 1932):

1. The lessee shall within 6 months of the date of the lease commence mining operations on the area leased and thereafter continuously and vigorously prosecute the work of mining.

2. The lessee shall keep continuously employed on the land the following number of laborers (or labor-saving apparatus equivalent thereto, at the rate of 1 hp. to 8 laborers):

Five for 2 acres or part thereof under a class A lease.

Five for every 20 acres or part thereof under a class B lease.

Five for every 40 acres or part thereof under a class C lease.

Transfer.--Except with the written consent of the Governor in Council, no lessee shall transfer a lease or any of the rights granted thereby. (Sec. 28, No. 7 of 1927.)

QUALIFICATIONS FOR APPLICANTS FOR PROSPECTING AND MINING PRIVILEGES

An applicant for a mining lease may be required to show that he commands sufficient working capital to ensure the proper development of the area applied for. The Governor may order that reports made by prospectors or engineers upon that matter be submitted to him. (Sec. 25, No. 7 of 1927.)

An applicant for a mining right or an exclusive prospecting license likewise may be required to prove that he has sufficient capital to cover working operations and any compensation that may be payable; and he may be required to furnish a financial guarantee, in a sum not to exceed \$1,000 a square mile,

to be given within 6 weeks (or any specified longer period) from the date of notification of the approval of the application. Failure to pay the guarantee may cause refusal of the application. (Secs. 10 and 19, No. 7 of 1927; sec. 12, No. 3 of 1932.)

FORFEITURE

Prospecting right, exclusive prospecting license, and mining right.--For breach of any of the provisions of the mining ordinance and regulations, by the holder of a prospecting right or license or of a mining right or by his attorney, agent, or servant, the Governor may summarily revoke the right or license in case the holder thereof fails within a specified time to show cause why the right or license should not be revoked. (Secs. 17 and 22, No. 7 of 1927.)

Mining lease.--The Governor in Council may terminate a lease (or in lieu thereof levy a penal rent, not to exceed three times the rent payable under the lease, in addition to the rent under the lease) upon the following conditions:

If the lessee shall commit a breach of the terms of the lease or of the provisions of the mining law and regulations and shall not make good the breach within such period (not less than 1 month) as the Governor may designate.

If the lessee shall wholly discontinue operations under the lease during a continuous period of 6 months without the written consent of the Governor in Council.

Publication of a revocation in the Gazette shall be considered sufficient notification. (Sec. 37, No. 7 of 1927.)

A lessee that has paid all rents, royalties, and other sums due under his lease may, within 1 month after the expiration or other termination of his lease, remove all or any of his plant, buildings, or other property. (Sec. 36, No. 7 of 1927.)

SURVEYS, BOUNDARIES, BEACONS, ETC.

Applicants for prospecting licenses and mining leases are required, before making applications, to erect beacons, according to specific directions. (Secs. 8 and 34, No. 3 of 1932.)

The Governor in Council may require, with respect to the land included in an application for an exclusive prospecting license or for a mining lease, that either a Government surveyor or a certified surveyor make a survey, at the expense of the applicant. (Sec. 44, No. 7 of 1927; secs. 9, 10, and 35, No. 3 of 1932.)

A surveyor when appointed shall notify the applicant to appear in order to identify the area beacons by him. The plan made by the surveyor shall show the boundaries, boundary marks, and the topographical features lying on the boundaries and in the case of a mining lease all topographical features in the area inclosed by the boundaries. The area as finally marked and beacons under the direction of the surveyor shall conform as nearly as possible with the area beacons by the applicant; but the ruling of the surveyor in these matters shall be final. (Secs. 11 and 35, No. 3 of 1932.)

Boundary beacons shall be constructed of concrete or such material as the Government Inspector shall approve. Further provisions with respect to the construction, numbering, and maintenance of beacons are made in sections 14, 15, 37, and 92, No. 3 of 1932.

With respect to a mining right, before application is made, the applicant shall erect temporary beacons to mark the highest and lowest points of the stream; when the application has been granted, the applicant shall erect and maintain such beacons as the Government Inspector shall direct. No survey, or only such survey as is necessary in order to measure the length of the course of the stream applied for, shall be required. Reasonable charges may be made by the Government Department in examining and measuring the area applied for. These charges, which shall be paid at the time of the examination, shall not be refunded if the application is refused. (Secs. 14, 20, 21, and 22, No. 3 of 1932.)

WATER RIGHTS

The Governor may, on such terms as he thinks proper, grant to a mining lessee a license to obtain and convey from any river, stream, or watercourse outside the area of his lease such volume of water as may be necessary for mining purposes and to occupy such land, Crown or private, as may be necessary in connection with pumping and conveying. (Sec. 41, No. 7 of 1927.) The rent payable for such land shall be determined by the Governor in Council. (Subsec. 2, sec. 31, No. 7 of 1927.)

The holder of a water right shall construct and maintain to the satisfaction of the Director of Public Works bridges at all points where his watercourse crosses road or path. (Sec. 78, No. 3 of 1932.)

An application for a water right, to be sent in duplicate to the Government Inspector of Mines, shall be accompanied by a plan, in duplicate, which shall show the proposed watercourse from the point of intake to the point of entry into the applicant's leased area, and also the site of any dam, reservoir, or pumping station on ground not in the leased area. (Sec. 78, No. 3 of 1932.)

Prohibitions against the unauthorized alteration of water supplies and against pollution of river, stream, or watercourse and other details in connection with water rights and their revocation are found in sections 38 to 43 of No. 7 of 1927.

RIGHTS OF OWNERS AND OCCUPIERS OF LAND

The owner of private land or the occupier of Crown land under a lease, location ticket, or permit to occupy is entitled to a notice from the holder of a prospecting right, exclusive prospecting license, or mining lease, in writing if the person to be notified is literate or through the Government Inspector of Mines if he is not able to read.

In the case of the holder of a prospecting privilege (right or license), the owner or occupier may require the prospector to give security by depositing with the Government such sums (not to exceed \$250) as the Governor in Council may direct for the payment of damages to surface rights or for damage to lands, trees, or crops. The owner or occupier may cause cessation of prospecting until such security shall have been given. In the case of mining privileges, when notice has been given of the existence of a lease, the owner or occupier shall state the amount of annual rent desired. Compensation likewise shall be payable to the owner or occupier by a mining lessee. (Secs. 9, 10, 11, 16, 19, 20, 22, 31, and 33, No. 7 of 1927; secs. 6 and 86, No. 3 of 1932.)

Should the Governor because of the existence of a mining right or lease revoke a location ticket or resume possession of lands occupied under a Crown lease or permit to occupy, the holder of the mining right or lease shall recompense the Government for the compensation paid to the occupier of the Crown land revoked or resumed, as the case may be. (Secs. 22 and 35, No. 7 of 1927.)

If, after the granting of a mining right or lease, a Crown lease, location ticket, or permit to occupy shall be granted for land within the area of a mining lease, the Crown lessee or other occupier shall not be entitled to compensation. (Sec. 35, No. 7 of 1927.)

Compensation is payable to the holder of a prospecting license, mining right, or mining lease in case his privilege is determined in favor of the holder of an oil license or lease or in case the privilege is determined in favor of the Government. Any dispute over the amount of compensation shall be determined by arbitration. (Sec. 54, No. 7 of 1927.)

RENTS, ROYALTIES, ETC.

Each mining lessee shall pay, in addition to royalties and rents payable for surface rights, the following rents:

Class A lease	\$2.50 an acre or part thereof
Class B lease	1.25 an acre or part thereof
Class C lease	0.25 an acre or part thereof
Classes D, E, and F	(as later to be fixed by regulations)

The Governor may, for good reason, reduce a rent temporarily or for the remainder of the term of the lease. (Sec. 29, No. 3 of 1932.)

Specific royalties for specific minerals are imposed: on a sliding scale for some ores, as tin and tungsten; at a certain percentage of their value for other ores or minerals, as lead, iron, mica, and precious metals other than gold; at a percentage to be determined by the Governor for gold--the rate not to exceed \$1 an ounce (troy).

If a mineral is to be exported, the Governor may permit the royalty due to be collected in the form of an export duty, payable upon exportation at the customs station.

With respect to tin ores, an export duty equal to 50 percent of the maximum royalty payable thereon may be charged, but this duty may be waived if the exporter shall have satisfied the Governor that all the tin exported by him will be smelted in British territory. (Secs. 1 to 4, No. 3 of 1932.) (See also secs. 20 and 31, No. 7 of 1927, and 38 and 84, No. 3 of 1932.)

Miscellaneous fees may be found in section 1 and in schedule 2, No. 3 of 1932. Detailed survey charges are listed in sections 11 and 36, No. 3 of 1932.

MISCELLANEOUS

Penalties

Any person committing a breach of any of the mining regulations for which a specific penalty is not provided shall, in addition to any liability to forfeiture under the ordinance (No. 3 of 1932), be liable to a fine of \$250 or in default thereof to imprisonment for 6 months. (Sec. 88, No. 3 of 1932.)

References to specific penalties are the following sections of the general mining law (No. 7 of 1927): 3, 5, 6, 9, 14, 26, 37, 39, 40, 52, 53, 56, 58, 64, and 68.

Disputes, Arbitration, and Appeals

The holder of a mining or prospecting privilege that considers himself injured by the mining or prospecting operations of any other person, by giving 14 days' notice to the other party may submit the matter through the Government Inspector to the Governor in Council, whose decision shall be final unless he himself shall direct that the case be decided by arbitration or in the courts. (Sec. 61, No. 7 of 1927.)

Arbitration proceedings shall be conducted under the arbitration ordinance, 1926, or under any statutory modification or reenactment thereof for the time being in force, with the following additions (sec. 63, No. 7 of 1927):

1. Unless the parties shall otherwise agree in writing, the reference shall be to two arbitrators, one to be appointed by each side, and the arbitrators may, if necessary, appoint an umpire.

2. When requested by one of the parties to an arbitration, the Governor in Council may appoint an arbitrator on behalf of such party and in that case may determine the fee to be paid to the arbitrator, or if the arbitrator is in the service of the Government the fee for his services.

In the case of an appeal to the Governor under any provision of the mining law and regulations, the Governor may require an affidavit of the facts affecting the case under appeal from any or all of the parties interested and from any officer of the Mines or Survey Department that is cognizant of the facts. (Sec. 91, No. 3 of 1932.)

Registration

Every mining lease, mining right, exclusive prospecting license, and water right, as well as every instrument by or under which the rights, or any portion thereof, shall be transferred, assigned, surrendered, or revoked, shall be recorded in the General Registry, within 1 month after the date thereof if executed in the colony and within 3 months if executed outside the colony, provided that the Chief Justice may in any particular case extend the time for recording a document. (Sec. 65, No. 7 of 1927; sec. 3, recording of deeds ordinance.)⁷

The Government Inspector of Mines shall keep a register of every mining lease, mining right, exclusive prospecting license, and water right, as well as every document assigning or transferring them or any right, title, or interest thereunder; he shall allow searches in the register at all reasonable times and shall furnish copies of or extracts from any entry in the register, if so requested. Details of registration are found in section 89, No. 3 of 1932.

Possession and Purchase of Minerals⁸

No person shall possess any reserved mineral unless he is the lessee of a mining lease, or the holder of a mining right, an exclusive prospecting license, or a prospecting right, or the holder of a license to purchase minerals (issued by the District Commissioner), or the duly authorized employee of such a lessee or holder. (Secs. 47 and 49, No. 7 of 1927.)

7 - Recording of deeds ordinance, Feb. 18, 1857: The New Edition of the Consolidated Laws of British Honduras, 1924, chap. 214, vol. 2, London, 1924, pp. 1617-1625.

8 - For the purpose of this section, "mineral" means any mineral that the Governor may, by notice in the Gazette, include under these provisions. (Sec. 46, No. 7 of 1927.)

No person shall purchase any mineral unless he is the holder of a license granted by the District Commissioner; and no holder of a license to purchase shall purchase any minerals until he is satisfied that the seller is authorized to be in possession of the mineral. (Secs. 48 and 50, No. 7 of 1927.)

Other provisions will be found in sections 46 to 52, inclusive, No. 7 of 1927, and in section 90, No. 3 of 1932.

Mining Regulations

The following matters relating to the regulation of actual mining operations are covered by the general mining ordinance and the corresponding regulations:

Liability of employer (sec. 62, No. 7 of 1927).

Labor provisions (sec. 82, No. 3 of 1932).

Government inspection and safety measures (secs. 59 and 60, No. 7 of 1927; sec. 83 and 87, No. 3 of 1932).

Plans, accounts, monthly and yearly returns (secs. 80 and 81, No. 3 of 1932).

Machinery (secs. 39 to 45, No. 3 of 1932).

Explosives (secs. 46 to 53, No. 3 of 1932).

Open-cast working (secs. 54 and 55, No. 3 of 1932).

Underground mining (secs. 56 to 76, No. 3 of 1932).

Tailings and deleterious substances (sec. 77, No. 3 of 1932).

PART III. - OIL

GENERAL

As prospecting and mining privileges with respect to oil in British Honduras are confined to British subjects and to British corporations (see section of this paper entitled "Rights of Foreigners"); as the actual provisions of the oil law itself (No. 32 of 1920) and of the regulations (of January 5, 1921) are limited; and as most of the conditions attending such privileges are included in the license and lease agreements, the form of which may be changed by the Governor from time to time, the provisions of the law and the regulations (relating principally to prospecting licenses and mining leases) are given in section 1 of this digest, and the more important provisions of the prescribed forms for prospecting and mining instruments are outlined in section 2.

Section 54 of No. 7 of 1927 (the general minerals law) gives the holder of an oil prospecting license or mining lease preference over the rights of the holders of privileges with respect to other minerals. The holder of oil privileges may request the revocation of other privileges under the general minerals law; or the Government, desiring to exercise oil rights, may terminate general licenses and leases for other minerals.

The Governor is given broad powers at any time to preempt all oil and products thereof and in case of war or other State emergency to assume entire control of the plant and premises of a lessee of oil rights.

A lessee is under obligation, if so required, to erect a refinery, which shall be capable of treating a named percentage of the output of oil from the leased area, and which shall be capable of producing residues suitable for the requirements of the Imperial Government. With the Governor's consent, the lessee may make arrangements with an approved refinery; or the Governor may waive the requirements for a fixed period or until a reasonable time after the Imperial Government shall have announced its readiness to purchase from the lessee stated quantities of oil residues at current prices. The site of the refinery, as well as of storage places, is subject to the Governor's approval. (Art. 10, reg. of 1921.)

PROVISIONS OF THE LAW AND THE REGULATIONS

Oil operations in British Honduras are carried on under exploration licenses, prospecting licenses, and mining leases. An exploration license is acquired only by a person desirous of obtaining a prospecting license; and a mining lease is granted only to a prospecting licensee that has given satisfactory evidence of having discovered crude oil or bituminous mineral on the licensed land. (Art. 5, No. 32 of 1920; arts. 1 and 3, reg. of 1921.)

Exploration License

An exclusive exploring license covers the right of the licensee to "explore, geologically examine and search," the surface of the land (that is, Crown land or land, whether under lease or not, upon which oil rights have been reserved to the Crown), to a depth not to exceed 20 feet, for petroleum oil and other bituminous minerals.⁹

Application.--An application for an exploring license shall be made to the Surveyor General, addressed to the Governor, describing the tract of land and giving the approximate acreage and boundaries thereof, together with full particulars as to the identity, name, address, and occupation or business of the person, persons, or company making the application. It must be accompanied by a deposit of \$10, and it is granted subject to the payment by the licensee of all damages to the land in question.

Duration.--No exploration license shall be issued for more than 2 years. (Art. 3, No. 32 of 1920; art. 1, reg. of 1921.)

Prospecting License

The conditions governing licenses to prospect upon land alienated subsequent to December 20, 1907 (art. 6-8, reg. of 1921), are practically the same as those governing licenses to prospect upon Crown land, except that a licensee of alienated land is subject to the payment of compensation to the owner of the land and to such "rates and duties" as may be imposed by law upon such produce, whereas the licensee of Crown land is subject to the payment of such "royalties, notes, and duties" as may be imposed by law. In each case, the licensee, if so required, shall give reasonable security that the actual development work agreed upon shall be undertaken during the currency of the license.

A prospecting license gives an exclusive privilege to search for crude oil, to set up works, to sink wells, and to export the produce.

Application.--An application for a prospecting right shall be addressed to the Governor and shall be transmitted to the Surveyor General. It shall describe the tract, give the approximate acreage, and set out the boundaries of the land to be prospected.

Fees.--A fee of \$25, to cover the Government expenses in connection with the petition, shall be deposited; and unless the area has been surveyed, a fee of 12½ cents an acre on the estimated acreage shall be paid.

Area.--The tract of land to be prospected shall not, except in special cases determined by the Governor, exceed one fourth of the area described in the exploration license. The area shall be portioned off in such a way as to meet the approval of the Governor.

9 - Bituminous mineral means any substance other than crude oil containing petroleum oil or any product thereof.

Duration.--A license is issued for 2 years, with the right of renewal for another term not to exceed 2 years, at the discretion of the Governor. (Art. 4, No. 32 of 1920; art. 2, reg. of 1921.)

Survey.--The Governor may, if he thinks fit, order any Crown lands applied for or under a search license to be surveyed. (Art. 11, reg. of 1921.)

Transfer.--Prospecting licenses are not transferable without the written consent of the Governor. (Art. 12, reg. of 1921.)

Mining Lease

Leases with respect to oil-bearing lands shall be subject to the approval of the Secretary of State. (Art. 10, reg. of 1921.)

A prospecting licensee, if he can prove that he has discovered crude oil or bituminous mineral substance, may with the Governor's consent lease, at an agreed annual rent, as much of the land as may be required or may be considered necessary by the Governor for the erection of works and buildings; and the licensee shall have the privilege of leasing the crude oil underlying or upon such portion of the licensed land as he may select. (Art. 5, No. 32 of 1920; art. 3, reg. of 1921.)

Duration.--A lease is granted for a period of 21 years or for such longer period as the Governor may approve; renewal may be had for like periods, provided the lessee has given to the Governor a 6 months' previous notice in writing, and provided he has paid the rent and royalties required and has observed other required covenants. The renewal shall be subject to the same covenants, provisions, and agreements as the original lease, save the provision for extension, and to such other conditions as the Governor may approve. (Arts. 3 and 4, reg. of 1921.)

Area.--The size of the area is subject to the applicant's desire and the Governor's discretion, but is limited, of course, to the prospected area. (Art. 3, reg. of 1921.)

Fees.--The applicant shall deposit \$25 with the Surveyor General to cover the Government's expenses in connection with the lease. (Art. 3, reg. of 1921.) All fees and expenses in connection with survey, grant, or registration shall be paid by the applicant. (Art. 11, reg. of 1921.)

Survey.--All lands to be leased must be first surveyed. (Art. 11, reg. of 1921.)

Transfer.--Leases are not transferable without the written consent of the Governor. (Art. 12, reg. of 1921.)

Forfeiture.--The lease shall become null and void if mining operations shall have been abandoned or discontinued for 1 year. (Art. 3, reg. of 1921.)

PRINCIPAL PROVISIONS PRESENTED IN LICENSE AND LEASE FORMS

Powers of the Governor

The powers of the Governor, as set forth in prescribed lease agreements, include, among others, the following.

Right to search or to authorize another to search for and mine minerals other than oil and natural gas upon oil-leased land.

Right to grant or demise to any person all or part of oil-leased lands for any purpose, subject to the rights of the oil lessee.

Right to make and maintain roads, tramways, railways, etc.

Right to preempt oil and products thereof; to require the production of oil fuel of the consistency, description, and flash point set out in the prescribed specifications; to order a lessee during an emergency to increase the production of oil; or during war or other State emergency to take entire control of a lessee's plant and premises.

Right to distrain crude oil, refined products, engines, plant, live and dead stock, etc., should a lessee fail to pay rent and royalty for two calendar months; right to reenter oil-leased land should a lessee fail to pay rent and royalty for 6 months.

The powers of the Governor, as set forth in prescribed license agreements, include, among others, the following:

To enter upon and to grant or demise to another liberty to enter upon licensed land for any purpose, and especially for the purpose of making roads, tramways, railways, and to obtain for that purpose from the land the necessary stones, earth, timber, or other material.

To declare summarily that a license shall be forfeited, should the licensee at any time refuse or neglect to observe any of the terms, and conditions of the license.

Rights of the Lessee

The privileges to be enjoyed by the lessee include, among others, the following:

To appropriate water, land for stock, etc., timber, and gravel, for use on the premises only.

To erect houses for workmen and sheds, engines, machinery, furnaces, pipe lines, railroads, tramroads, etc., necessary for mining operations.

To refine oil for use on the land or for sale; to store, take, lead, pipe, and carry away on, under, or over the land crude oil and refined products.

To terminate the lease should the premises become unfit for the purposes thereof through circumstances beyond the control of the lessee or of his servants or to surrender the lease by giving 6 months' notice should the borings or works become unprofitable.

Rights of the Licensee

The ancillary privileges to be enjoyed by a licensee include, among others, the following:

To clear away brushwood.

To make (with the written consent of the Governor) use of drains or watercourses.

To build all necessary roads.

To erect temporary sheds and other structures and to bring upon the land steam and other engines, machinery, conveniences, chattels, and effects.

Arbitration

: Disputes, in general, under both licenses and leases, shall be determined by arbitration under the provisions of the Arbitration Act of 1889 of the Imperial Parliament or of any statutory modification thereof.

The license agreement makes the following stipulation:

Provided always that nothing herein contained shall extend to or include any question or dispute that may arise as to the exercise by the Governor of the powers and rights summarily to determine the license.

The lease agreement excludes from arbitration any matters expressly agreed to or intended to be determined by the Governor or by the lessee.

Bonds

A licensee, before he shall be granted an exploring license, shall enter into a bond with the Colonial Treasurer for an amount calculated on the basis of $1\frac{1}{2}$ cents an acre of the area applied for, conditioned upon his expending a like amount on exploring work; and noncompliance with this condition shall entail forfeiture of the bond as to the whole or a part thereof, as the Governor in Council may decide. A like bond must be executed in case of an extension of the license.

Similar bonds, in an amount to be stated therein ("by way of liquidated damages"), are executed by prospecting licensees and by mining lessees, as surety that they will perform all the covenants of their deeds (licenses or leases).

Working Requirements

The lease agreement sets forth the maximum period of time that may elapse before work shall have been begun; states that a minimum of one rig shall be employed on a given number of acres; stipulates the number of feet that shall be drilled in any year or the total output in tons of crude oil that shall be raised; and requires that from time to time additional wells to a given depth per annum shall be drilled or that wells of less depth shall be drilled in the number necessary to maintain the required output per annum.

The prospecting-license agreement provides that work shall be begun with all reasonable dispatch, stipulates the number of wells (and the aggregate depth thereof) that must be sunk, with the proviso that the Governor may dispense with further prospecting whenever drilling results justify the issue of a mining lease.

A licensee shall provide as a minimum one rig with a suitable number of bore-masters and appliances for a given number of acres.

Royalty and Rent

Royalty under a prospecting license.--Schedule B attached to the agreement form for a prospecting license makes the following provision concerning royalty:

For all crude oil a royalty of 50 cents a ton shall be paid to the Colonial Treasury on the net output collected into the field tanks of the licensee after the deduction of water and foreign substances (except such quantity as is herein granted free of royalty); and a royalty of 2 cents shall be paid for every 1,000 cubic feet of natural gas (calculated at an absolute pressure of 1 atmosphere and a temperature of 60° F.) sold by the licensee.

The Governor in Council may revise at intervals during the currency of the license the rates of royalties aforesaid, in accordance with the fluctuations in the price of oil.

Rent and royalty under a mining lease.--Part 5 of the lease form provides for the payment by the lessee of a certain fixed half-yearly rent (to be specified in each lease). It provides a further yearly rent of 12½ cents an acre or part thereof, to be paid in equal half-yearly installments.

Part 5 of the lease form contains with respect to royalties the following provision:

From the commencement of the term granted there shall be payable to the Treasury, half-yearly, a royalty of 50 cents a ton of net crude oil that shall have been received into the lessee's field storage tanks after deduction of water and foreign substances and oil used for carrying on the works and for domestic consumption in the houses and offices of agents and workmen, and a royalty of 2 cents per 1,000 cubic feet of natural gas (calculated at an absolute pressure of 1 atmosphere and a temperature of 60° F.) sold by the lessee.

Stipulations in the lease concerning rent and royalty are as follows:

If the total royalties upon crude oil in any half year shall amount to or exceed a named sum, the certain half-yearly rent shall not be payable.

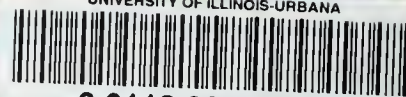
If the total royalties shall be less than the named sum, the amount of the royalties shall be deducted from the certain half-yearly rent.

If the quantity of net crude oil received into the lessee's field storage tanks during any complete year exceeds 100,000 tons, a deduction of 12 cents a ton shall be made from the royalty for every ton so received during the year, but so that at least the amount of the certain rent shall be paid with respect to each and every half year.

If in any half year the lessee shall not win or get from his land such quantity of crude oil as, at the rates mentioned in the preceding paragraphs, would produce for that half year the certain rent, then the lessee may in the next succeeding half year (but not afterwards) win from the borings such a quantity as may be necessary to make up the deficiency without paying royalty therefor (surplus of preceding half years may not be applied).

If the Governor calls for the delivery of 10 percent of the net crude production (as he has legal right to do), the lessee shall not pay royalty on the quantity taken or a further quantity equal to nine times the quantity taken. In addition, the certain half-yearly rent shall, for the half year in which the oil was taken, be reduced by a sum equal to \$5 a ton for the oil taken by the Governor or \$4 if the quantity of net crude oil received into the lessee's field storage tanks in the complete year exceeds 133,333 tons.

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